Audit 2014
Volume 7 Mollakara Resources and Reserves
Koza Altın İşletmeleri A.Ş.
Turkey

Report Prepared for

Koza Altın İşletmeleri A.Ş.

Report Prepared by

SRK Consulting (U.S.), Inc.
SRK Project Number 173600.130
January 31, 2015

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Table of Contents

List of Tables

List of Figures

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List of Abbreviations

The metric system has been used throughout this report unless otherwise stated. All currency is in U.S. dollars unless stated otherwise. Market prices are reported in US$ per troy oz of gold and silver. Tonnes are metric of 1,000 kg, or 2,204.6 lb, unless otherwise stated. The following abbreviations are typical to the mining industry and may be used in this report.

Abbreviation	Unit or Term
°	degree
%	percent
AA	atomic absorption
AAS	atomic absorption spectography
Ag	silver
Amsl	above mean sea level
Au	gold
BLEG	Bulk Leach Extractible Gold
BWI	Bond Work Index
C	Celsius
CoG	cutoff grade
CIP	carbon in pulp
cm	centimeter
CP	Competent Person
CPR	Competent Person's Report
CRP	Community Relations Plan
CRM	Certified Reference Material
Cu	copper
dia.	diameter
Eq	equivalent
EIA	Environmental Impact Assessment
F	Fahrenheit
ft	feet/foot
g	gram
g/cm	grams per centimeter
g/t	grams per tonne
ha	hectares
HG	high-grade
hr	hour
ID2	Inverse Distance Squared
ID3	Inverse Distance Cubed
in	inch
IP	Induced Polarization
kg	kilogram
km	kilometer
koz	thousand troy ounce
kt	thousand tonnes
kV	kilovolt
kVA	kilovolt-amps
L	liter
lb	pound
LHD	load haul dump
LG	low-grade
LoM	life of mine

m	meter
M	million
m.a.	million annum
min	minute
mm	millimeter
Mm	million meter
Moz	million ounces
Mt	million tonnes
Mt/y	million tonnes per year
MVA	million volts amperes
NN	Nearest Neighbor
NPV	net present value
OK	Ordinary Kriging
OP	open pit
oz	ounce
ppb	parts per billion
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
RC	reverse circulation
RoM	run of mine
SART	sulfidization, acidification, recycling, and thickening
t	tonne(s)
t/h	tonnes per hour
t/d	tonnes per day
t/m	tonnes per month
t/y	tonnes per year
TEM	Technical Economic Model
μ	micron
UG	underground
V	volt
WAD	weak acid dissociable
Zn	zinc

1 Introduction

SRK Consulting (U.S.), Inc. (SRK) was commissioned by Koza Altın İşletmeleri A.Ş. (Koza) to audit Koza's gold resources and reserves and exploration projects as of the end of December, 2014. Koza's Mining Assets are located in the Ovacık Mining District, Mastra Mining District, and Kaymaz District, including Söğüt, as well as Mollakara in the Diyadin District in Eastern Turkey and Himmetdede in Central Turkey.

This report is Volume 7 Mollakara Resources and Reserves of the following ten volumes reports:

- Volume 1 Executive Summary;
- Volume 2 Ovacık Resources and Reserves;
- Volume 3 Mastra Resources and Reserves;
- Volume 4 Kaymaz Resources and Reserves;
- Volume 5 Söğüt Resources and Reserves
- Volume 6 Himmetdede Resources and Reserves;
- **Volume 7 Mollakara Resources and Reserves**;
- Volume 8 Technical Economics;
- Volume 9 Hasandağ and Işıkdere Resource Areas; and
- Volume 10 Exploration Projects.

This report is prepared in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code 2012).

Volume I Executive Summary contains the Terms of Reference and Property Descriptions relevant to all volumes of this audit.

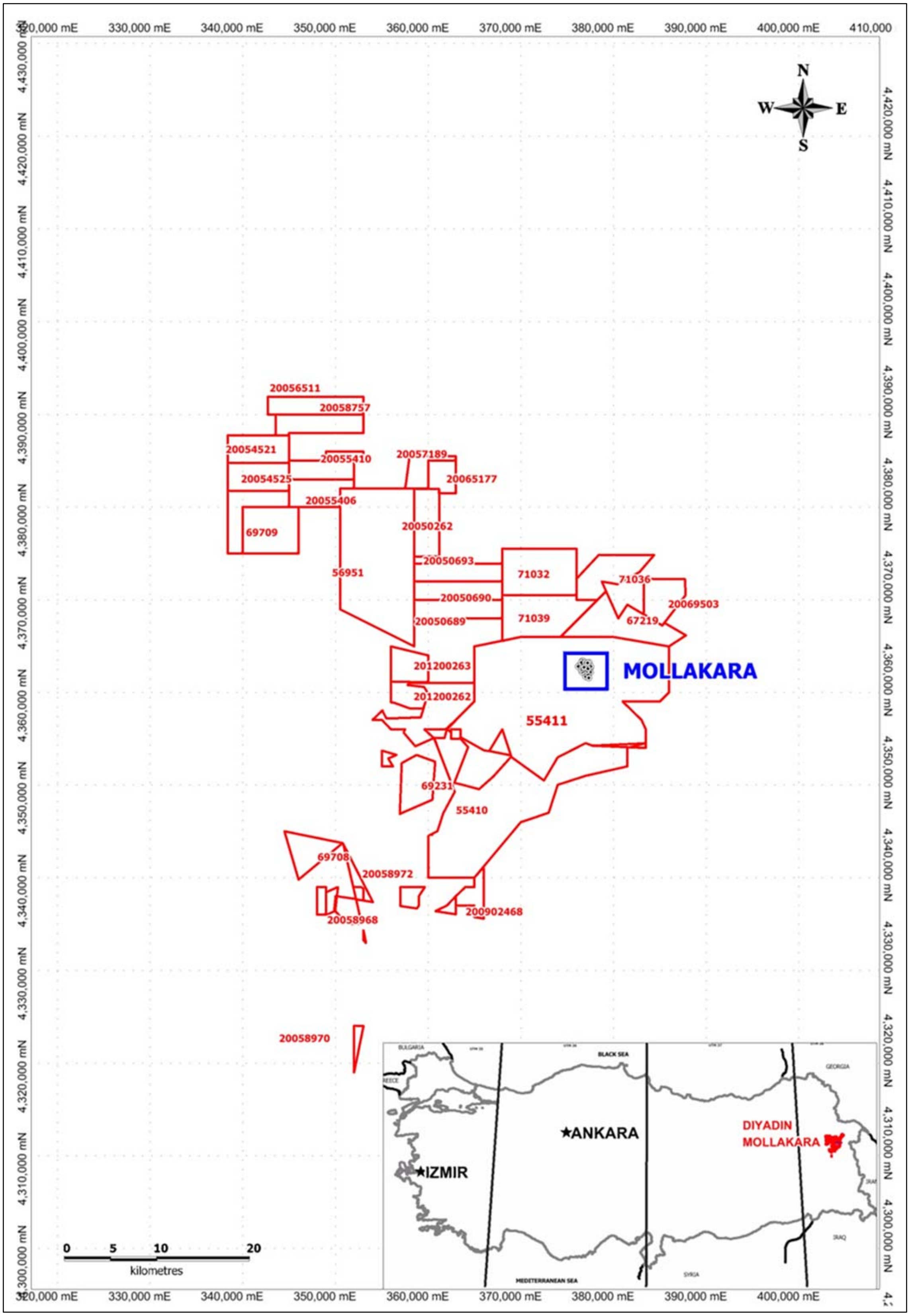
2 Mollakara Feasibility Project

2.1 Property Description and Location

The Mollakara Project (the Project) is located in Eastern Turkey in the Diyadin District, approximately 100 km north of Van, 200 km southeast of Erzurum and 920 km east of Ankara by air. Figure 2.1.1 shows the project location in Turkey and Figure 2.1.2 shows the project land tenure and location relative to Diyadin and other Koza projects and licenses in the Diyadin District.

Source: Modified from ESRI Basemaps NatGeo_World_Map, 2013

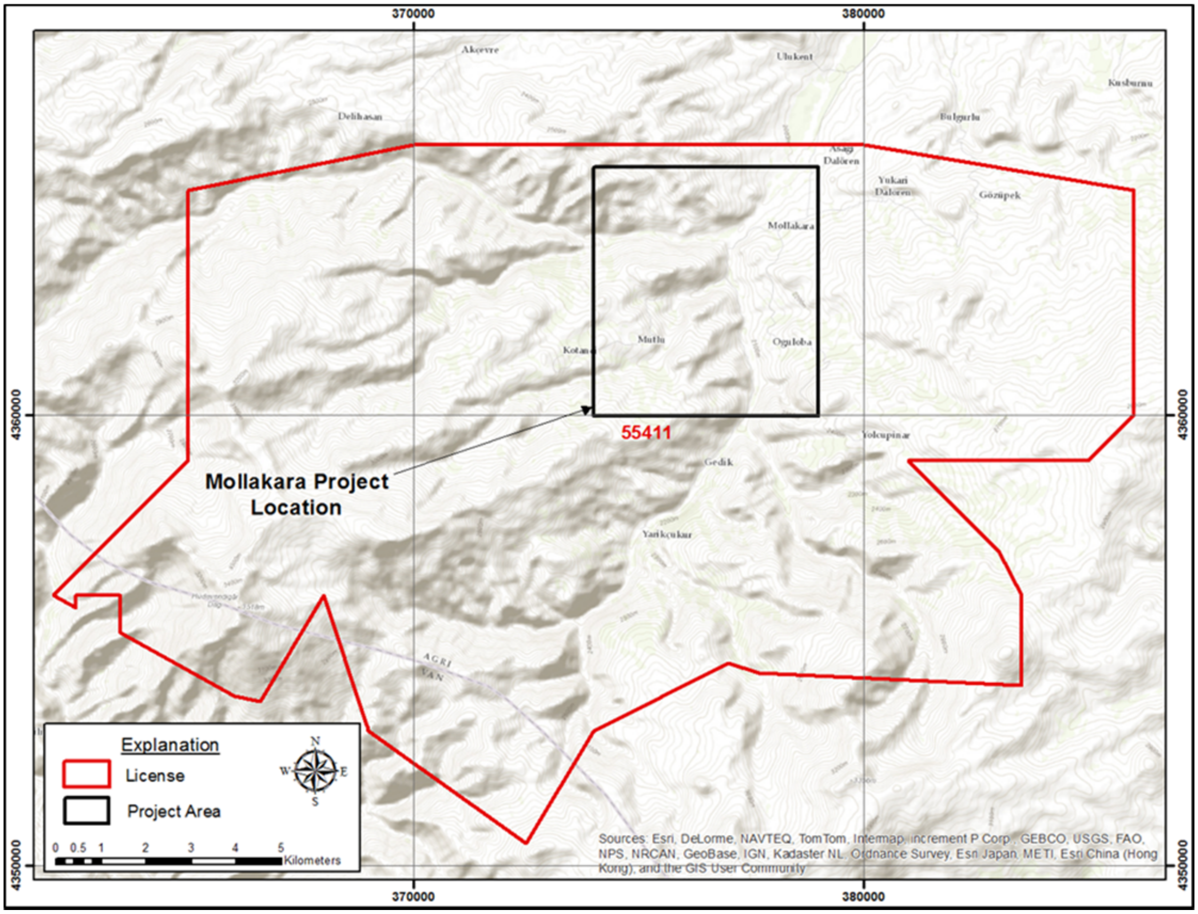
Figure 2.1.1: Mollakara Project Location Map



Source: Koza, 2013

Figure 2.1.2: Mollakara Project Location in the Diyadin District

The closest town to the project area is Ağri, located approximately 55 km northeast of Mollakara. Ağri is connected by international road E80 to Erzurum, which is located 180 km east and has a domestic airport. The project is accessed from Ağri by taking road E80 west for 50 km then south on road 04-26 for 7 km to Diyadin. From Diyadin, the Project and Mollakara village are 15 km south along unpaved roads. Mollakara is located between Universal Transverse Mercator (UTM) coordinates 4365500 N, 374000 E and 4360000 N, 379000 E European Datum 1950 (ED50) Zone 38. Mollakara is within operation license 55411 totaling approximately 24,460 ha. Land tenure for Mollakara is shown in Figure 2.1.3.



Source: Koza, 2012

Figure 2.1.3: Mollakara Project Land Tenure Map

2.2 Climate and Physiography

The Diyadin District is located in Eastern Anatolia and includes the Mollakara, Ağadeve, Çakillitepe, Küçükdoğutepe and Taşkapi Projects. These projects are located in a continental climate with slightly more precipitation than Central Anatolia. This region is subject to cold harsh winters and dry warm summers. At Van average temperatures range from -3.3°C in January to 21.1°C in July and August. Temperatures as high as 44°C and as low as -45°C have been recorded in Eastern Anatolia. Temperatures are slightly cooler at higher elevations. Total precipitation is approximately 570mm and falls as rain in the summer and snow in the winter. Mollakara is in an area of low to moderate relief at approximately 2,100 m amsl.

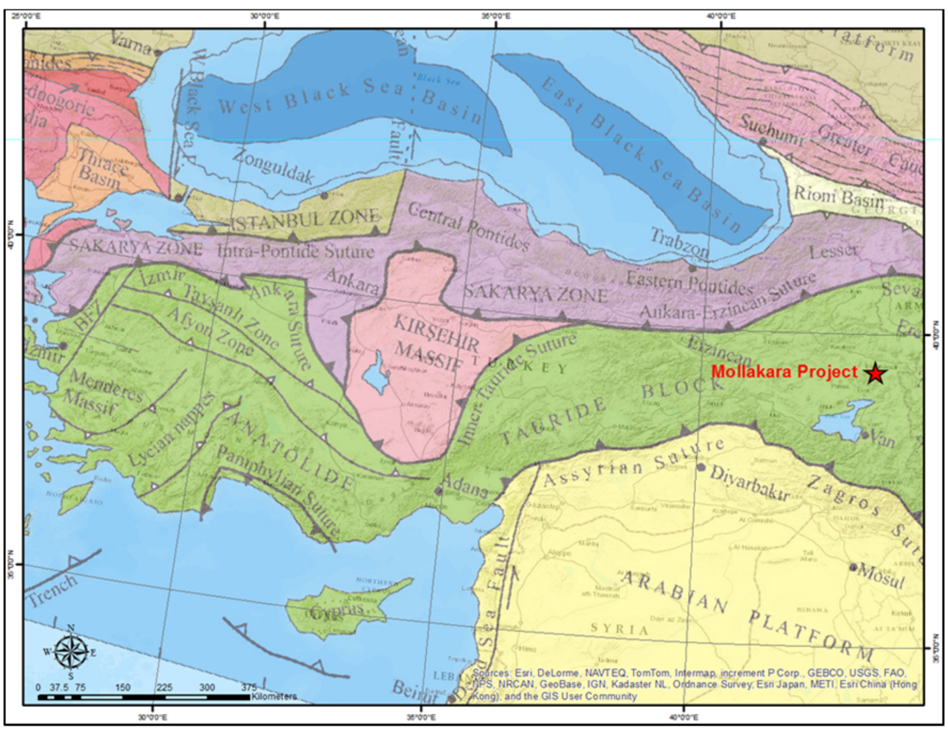
2.3 History

The Mollakara Project was held by Newmont Mining Corp. (Newmont) between 2005 and 2008. During that time, Newmont collected 20 Bulk Leach Extractible Gold (BLEG), 66 stream sediment, 2,063 soil and 1,551 rock chip samples. Newmont also mapped the project area at a 1:10,000 scale, completed 19 Reverse Circulation (RC) and 27 core holes and conducted an Induced Polarization (IP), resistivity survey at the project. Koza acquired the project in 2008 and since then, has drilled 101 core holes.

2.4 Geology

2.4.1 Regional Geology

The Diyadin District is located in eastern Turkey north of Lake Van in the Anatolide-Tauride block. The Anatolide-Tauride block is considered a single paleogeographic body or terrane bounded by the Ankara-Erzincan suture zone to the north and the Bitlis-Zagros suture to the south. This area of the Anatolide-Tauride block is the Turkish-Persian high plateau, which rises approximately 2 km amsl. This plateau formed as a result of Neotectonic convergence, collision and subsequent subduction of the Arabian Plate beneath the Eurasia Plate at the Bitlis-Zagros suture during the closing of the southern branch of the Neo-Tethyan Sea. The youngest unit associated with this closure is the Adilcevaz Limestone of Serravallian in age. Figure 2.4.1.1 shows the Mollakara project location in the Anatolide-Tauride block.



Source: Modified from Okay et al., 2010. Basemap = Basemaps NatGeo_World_Map, 2013

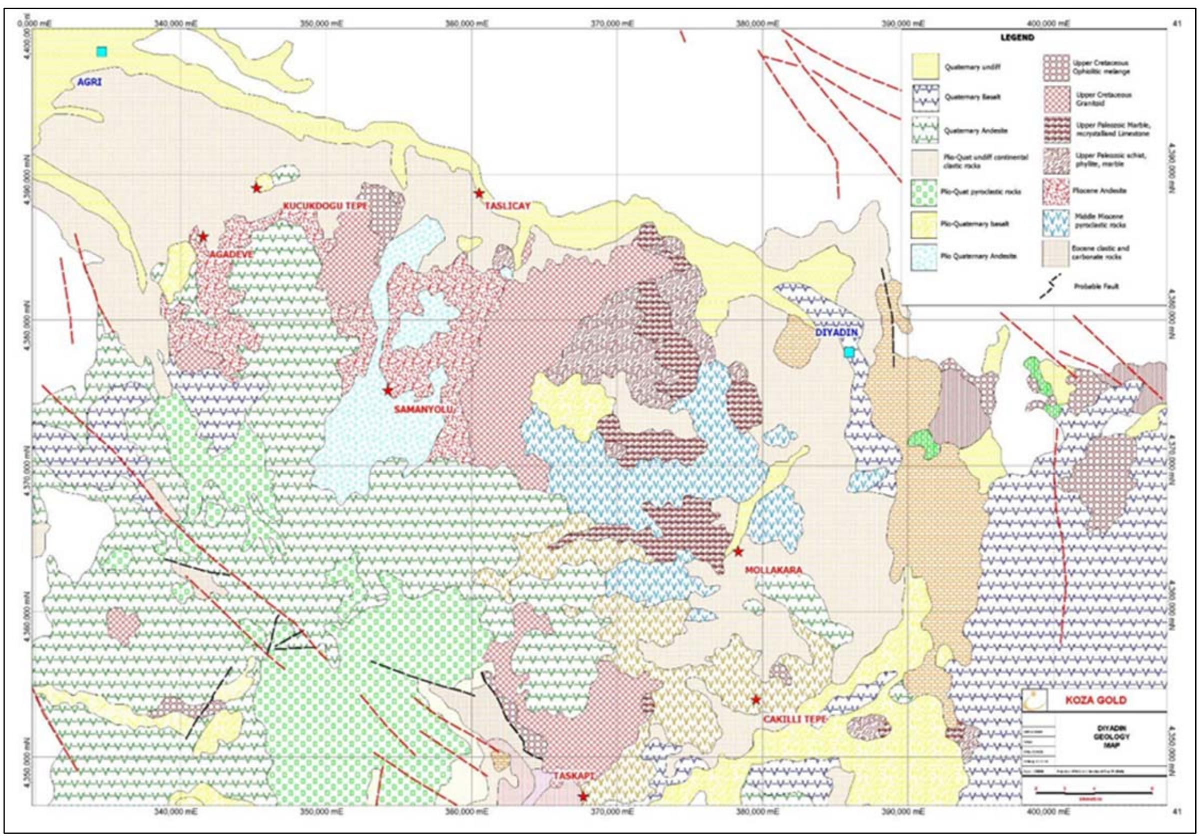
Figure 2.4.1.1: Location of the Mollakara Project in the Anatolide-Tauride Block

The Anatolide-Tauride block shows several different metamorphic, structural and stratigraphic zones throughout its extent. However, these zones share the following lithologic characteristics:

- Late Precambrian crystalline basement;
- Mixed clastic-carbonate Paleozoic succession; and
- Upper Triassic to Upper Cretaceous carbonate sequence.

Arc volcanism developed after the onset of subduction during the Late Miocene. This is identified by a N40°E trending line of volcanoes beginning at the west side of Lake Van, continuing along the northwest shore and extending toward the Turkish border. These volcanoes, named from southwest to northeast are Nemrut Dağı, Süphan Dağı, Tendürek Dağı, and Mount Ararat. North of this line, in the Diyadin District, the Anatolide-Tauride block is covered and intruded by volcanic rocks ranging in composition from rhyolite to basalt. Tendürek Dağı, located immediately east of the district, formed within the Balık Lake Fault zone. This is a strike-slip fault with a northwest-southeast strike. The volcanic rocks at Tendürek Dağı are highly alkaline and although the two craters line up along the northeast- southwest trend, the entire volcano is elongate parallel to the Balık Fault and trends toward Diyadin. The Balık Lake Fault has a similar northwest strike as the Çaldıran Fault.

Back arc volcanics and intrusions related to subduction along these fault zones have formed the hydrothermal drive and ground preparation for the formation of these deposits. The regional geology of the Diyadin District is shown in Figure 2.4.1.2.



Source: Koza, 2012

Figure 2.4.1.2: Regional Geology Map of the Diyadin District

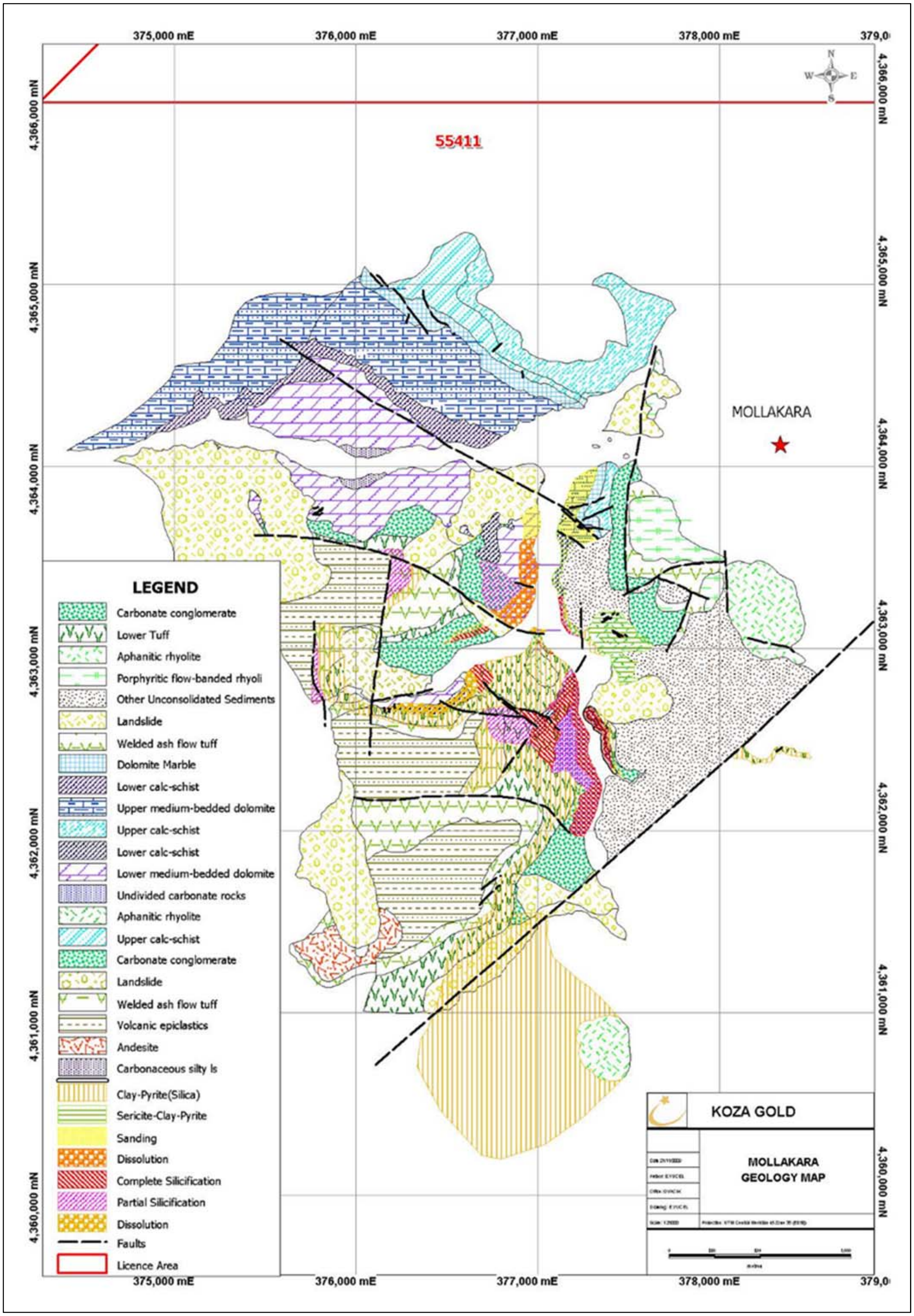
2.4.2 Local Geology

Mineralization at the Mollakara Project is interpreted by Koza as Carlin style mineralization and this exploration model is used by Koza within the project area. At the Mollakara Project, the rocks include a Paleozoic succession of carbonate rocks considered to be part of the Anatolide-Tauride block of marine basin origin. This sequence is capped by a Miocene age welded tuff (Figure 2.4.1.2). The carbonate rocks were deposited in a shelf and margin setting. During the Alpide orogeny and associated subduction, these rocks were folded with possible nappe and thrust development. Older carbonates have been folded into a doubly plunging anticline, one axis of which trends about N60°E, the other approximately N20°W. The half wave-length of these folds is on the order of 1 km, and the amplitude is approximately 100 to 200 m.

Within the mapped area, the defined stratigraphy has been treated as a simple, upright sequence. Koza has identified strain indicators that include stretch lineations in bedding planes, kink folding and isoclinal folding in individual beds, boudinage of sandy layers and sheath folds that indicate the rock has undergone a simple shear type deformation during folding. Koza is of the opinion that large-scale mapping of the carbonates may identify much larger scale deformational features such as thrust and nappe style tectonics.

Paleotopography of the older carbonates is abrupt and irregular indicating steep sided canyons up to 150 m deep. Carbonate conglomerate filled these canyons, and this formation is characterized by rapid lateral changes in facies and thickness which does not necessarily reflect faulting. High local erosion resulted in deposition of carbonate clast conglomerate in these topographic lows. The upper section of these conglomerates is interbedded with ashfall and ashflow tuffs. During the same period as the eruption of the tuffs and major volcanic activity, faulting is thought to have developed along the northeast, northwest and north-south fault sets seen in the project area. The northeast set of structures consists of normal faults striking N40-50°E which are found near the southern boundary of the mapped area. These have been interpreted as part of a regional structure with widespread argillization and silicification along this and similar striking structures. Koza interprets this set as important conduits for hydrothermal fluids. The N60-80°W faults are oblique-slip faults with an apparent left lateral slip component. Displacement on the faults is inferred to be up to 100 m. Generally, these faults appear to offset mineralization. The third set of faults strike N to N10°E and are high-angle normal faults. These faults are down-dropped to the west in the western part of the mapped area and down-dropped to the east in the eastern portion, forming a horst structure. The faulting has exposed silicified carbonates in the bottom of Murat Canyon adjacent to the Mollakara Project. Koza has mapped this faulting for a few hundred meters of strike length and interprets them as extensional faults linking the northeast and northwest striking oblique-slip faults described above. This structural zone appears to control the trend of Murat canyon and the distribution of hot springs along the canyon.

The stratigraphy of the project area includes a basal dolomitic marble overlain by alternating layers of dolomite and calc-schist capped by intermediate to acidic tuffs. The protolith of the calc-schist is thought to be a silty dolomite. Rhyolitic domes have been mapped in the project area outside of the mineralized zone. Project geology is shown in Figure 2.4.2.1.



Source: Koza, 2012

Figure 2.4.2.1: Mollakara Project Geology Map

Alteration identified in the carbonate rocks includes complete and partial silicification, dissolution of the host, sanding and calcite veining. Complete silica replacement is observed as jasperoid, breccias and porous textures. Partial silica replacement is observed in carbonate conglomerates and dissolution breccias which contain a silica matrix with carbonate clasts. Partial silica replacement also includes silica veining and in places hydrothermal breccias with crosscutting silica matrix. Dissolution results in cavity formation and calcite/travertine deposits, exceeding 5% to 10% of the rock. This includes intense dissolution breccia and can occur in thin-bedded units. Dissolution textures are found peripheral to, and north of, the jasperoid outcrops along Murat Creek. Sanding is the disintegration of granular dolomite as the result of the removal of calcite cement. Sanding is found peripheral to, and north of, the dissolution zone along Murat and Kendal Creeks, and around Mollakara Village. Both dissolution and sanding indicate moderate to strong acidic conditions during mineralization. Calcite veins are rare and Koza suggest that this may indicate that mapping has not yet reached the limits of the hydrothermal system where calcite would be expected.

Alteration in the volcanic rock also includes complete and partial replacement by silica. These alteration types are identified by porous textures resulting from leaching of clay material usually proximal to tuff capping jasperoids. Partial silicification also includes silica veining and silica replacement of matrix in crystal lithic tuffs with the pumice, phenocrysts, and lithic fragments exhibiting clay alteration. This can be difficult to identify in weathered outcrops.

Quartz-sericite-pyrite (QSP) alteration has also been mapped in the project area in two rock types: the platy aphanitic rhyolite mapped on the east side of Murat Creek and the calc-schist layers.

Mineralization at Mollakara is both structurally and lithologically controlled. Koza has not yet identified the feeder zones for mineralization at this project. Locating the feeder zone is currently a focus for exploration activities at Mollakara. Koza’s interpretation of the mineralization at Mollakara is:

- Alteration and structural patterns suggest that the Diyadin hydrothermal system was centered to the east or southeast of the jasperoid identified along Murat Creek, possibly at the juncture of northeast and north-south striking structures;
- Fluids ascended along north-south and northwest striking structures, and up east-dipping carbonate stratigraphy. Argillically altered tuffs acted as a cap rock, trapping fluids in the underlying carbonates;
- Thin bedded and silty (micaceous) carbonate rocks were preferentially replaced with silica and mineralized with Au;
- Overlying carbonate-clast conglomerates were also favorable units for mineralization while thick-bedded dolomites were partially replaced and generally have lower Au grades; and
- Fluids moved outward along north-south and northwest striking structures, producing an aureole of weaker acidic alteration including partial silicification, dissolution, sanding and moderate argillic alteration in volcanic rocks.

2.5 Exploration

Koza has not conducted any exploration at the Mollakara project since 2011. Future exploration will focus on:

- Structural feeders to the system, including potential north-south structures to the east of the jasperoids outcropping in Murat Creek;

- Down-dip extensions of favorable stratigraphy, including the lower calc-schist, the upper portion of the lower medium-bedded unit, and the underlying carbonaceous limestone; and
- Potential structural extensions, underlying argillically altered volcanics to the northwest, and north of the outcropping jasperoid.

Koza has budgeted approximately TL3.5 million (US$1.6 million) for the 2015 exploration program focusing primarily on drilling.

2.5.1 Drilling/Sampling Procedures

The Mollakara deposit was drilled by Newmont between 2005 and 2006 and by Koza in 2009 and 2010. Table 2.5.1.1 summarizes the drilling at Mollakara and Figure 2.5.1.1 is a drillhole location map. The drillholes are predominately vertical which is appropriate for the shallowly dipping mineralization at Mollakara.

Table 2.5.1.1: Mollakara Drillhole Statistics

Company	Core Holes		RC Holes		Core Samples		RC Samples	
	Number	Meters	Number	Meters	Number	Meters	Number	Meters
Newmont	19	4,490.40	27	2,994	2,507	4,374	2,996	2,993
Koza	106	11,116.20			10,943	10,846		
Total	**125**	**15,606.60**	**27**	**2,994**	**13,450**	**15,220**	**2,996**	**2,993**

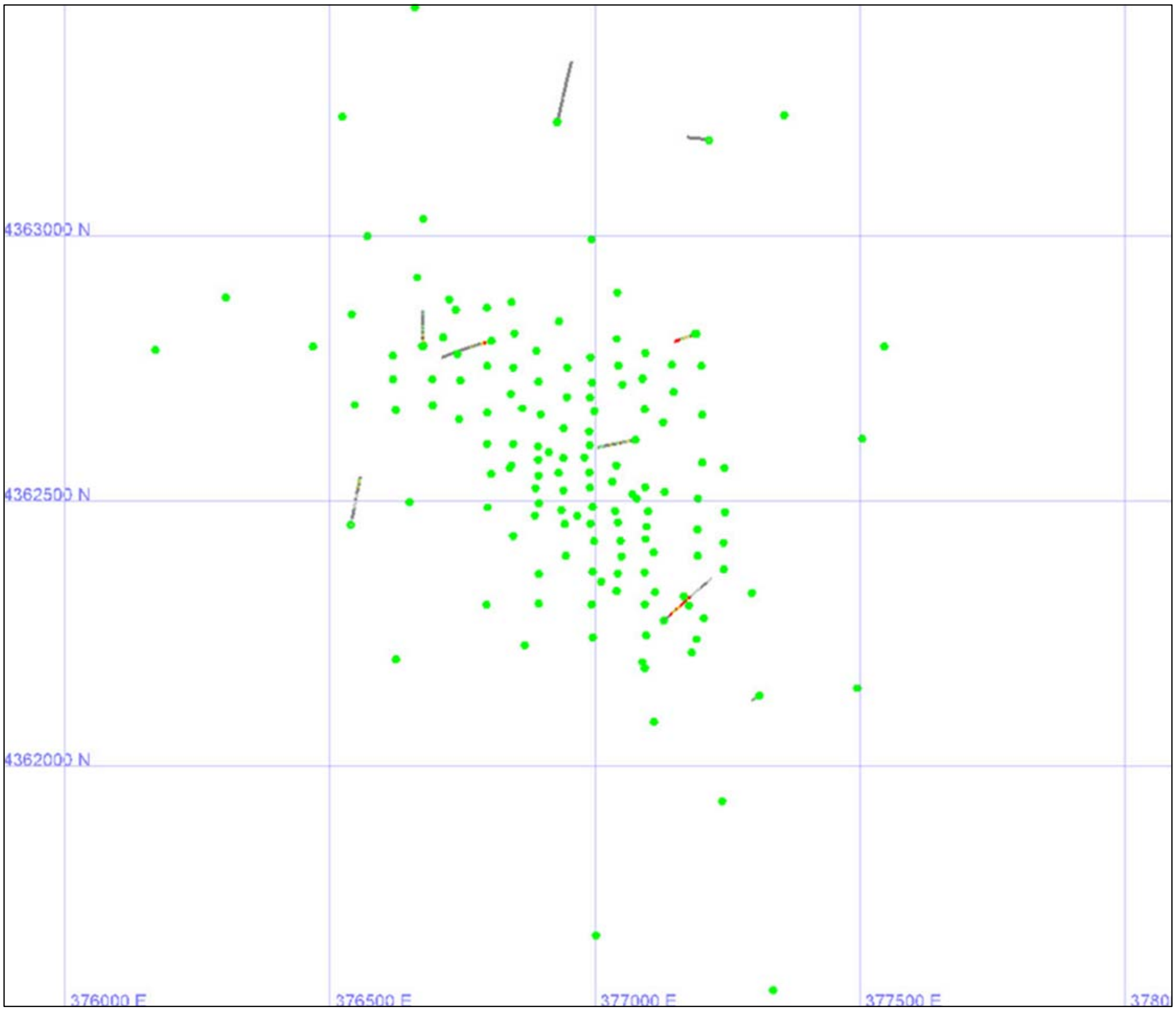


Figure 2.5.1.1: Mollakara Drillhole Location Map

Newmont conducted the first drilling program at Mollakara in 2005. A total of 27 RC holes (2,994 m) were drilled. It is reported (Koza, 2011) that the drilling encountered numerous problems including an inexperienced RC driller, hot water, sand and rubbly overburden. Sample recovery and quality varied considerably with suspected contamination and smearing occurring over wide intervals. Only seven of the 27 holes reached the target depth of 150 m. No downhole surveys were done on the RC holes. SRK has investigated four pseudo-twin pairs of RC and core holes that are each within 25 m of the other, and while there is considerable variation between the holes, there does not appear to be obvious contamination in the RC holes.

The RC drilling samples were collected in plastic bags at 1 m intervals. Splitting the samples proved problematic because of clay, sand and high water flows and therefore the final laboratory samples were collected by trowel after mixing in the plastic sample bag, or by spearing each bag twice diagonally using a PVC sample spear.

Newmont also drilled 19 core holes (4,490.4 m) in 2005 and 2006. The holes were drilled with HQ tools, reducing to NQ only when necessary due to drilling conditions. Four of the Newmont core holes were surveyed for downhole deviation.

Koza drilled an additional 106 HQ and PQ sized core holes for a total of 11,116.2 m between 2009 and 2011. The drillholes were surveyed for downhole deviation with a multi-shot camera after completion of the hole. Drill collar locations were surveyed by Koza surveyors.

The drill core was placed in wooden boxes, pieced together, measured and marked at 1 m intervals and photographed prior to geotechnical and geological logging. The sample intervals were marked by the geologist and the core was sawed lengthwise using a diamond saw. Sample preparation is conducted at ALS Turkey located in İzmir and the pulps are analyzed at ALS Romania for Fire Assay (FA) and ALS Canada for Inductively Coupled Plasma (ICP) analysis.

Once the samples arrived at the laboratory, they were bar coded and entered into the Laboratory Information Management System (LIMS). All samples were dried to a maximum temperature of 60°C in order to avoid or limit volatilization of elements such as mercury (ALS code DRY-22).

After drying, core samples were crushed to 70% passing -2 mm (ALS code CRU-31) and a 1,000 g split was collected using a riffle splitter (ALS code SPL-21). The 1,000 g split was pulverized to 85% passing 75 microns (ALS code PUL-32). Koza requests a larger split pulverized to help mitigate the nugget affect.

Core samples were analyzed using ALS code ME-ICP61m, a 33 element package with trace level sensitivity. A 1 g sample is put into solution using a four acid digestion and the sample is analyzed using ICP-AES. The package includes mercury analyzed by method Hg-CV41. In this method, mercury content is determined using aqua regia digestion and cold vapor AAS. Gold was analyzed using ALS code Au-AA24, which is gold by FA using a 50g charge with an Atomic Absorption Spectroscopy (AAS) finish. Table 2.5.5.2 presents the analytes with upper and lower detection limits for ALS ME-ICP61m, Hg-CV41 and Au-AA24.

Table 2.5.1.2: Analytes and Upper and Lower Detection Limits for ALS Codes ME-ICP61m, Hg-CV41 and Au-AA24 in ppm Unless Otherwise Noted

Method	Analyte	Range	Method	Analyte	Range	Method	Analyte	Range
Au-AA24	Au	0.005-10	ME-ICP61m	Cu	1-10,000	ME-ICP61m	S	0.01-10%
Hg-CV41	Hg	0.01-100	ME-ICP61m	Fe	0.01-50%	ME-ICP61m	Sb	5-10,000
ME-ICP61m	Ag	0.5-100	ME-ICP61m	Ga	10-10,000	ME-ICP61m	Sc	1-10,000
ME-ICP61m	Al	0.01-50%	ME-ICP61m	K	0.01-10%	ME-ICP61m	Sr	1-10,000
ME-ICP61m	As	5-10,000	ME-ICP61m	La	10-10,000	ME-ICP61m	Th	20-10,000
ME-ICP61m	Ba	10-10,000	ME-ICP61m	Mg	0.01-50%	ME-ICP61m	Ti	0.01-10%
ME-ICP61m	Be	0.5-1,000	ME-ICP61m	Mn	5-100,000	ME-ICP61m	Tl	10-10,000
ME-ICP61m	Bi	2-10,000	ME-ICP61m	Mo	1-10,000	ME-ICP61m	U	10-10,000
ME-ICP61m	Ca	0.01-50%	ME-ICP61m	Na	0.01-10%	ME-ICP61m	V	1-10,000
ME-ICP61m	Cd	0.05-1,000	ME-ICP61m	Ni	1-10,000	ME-ICP61m	W	10-10,000
ME-ICP61m	Co	1-10,000	ME-ICP61m	P	10-10,000	ME-ICP61m	Zn	2-10,000
ME-ICP61m	Cr	1-10,000	ME-ICP61m	Pb	2-10,000			

Source: ALS Global, 2014

Koza has a Quality Assurance/Quality Control (QA/QC) program independent of the laboratory. The QA/QC program includes certified reference material (CRM) blanks and core duplicates (quarter core). Koza inserts QA/QC samples at the following frequencies:

- A CRM every 30th sample;
- A blank every drillhole; and
- A core duplicate every 50th sample.

Should a QA/QC sample fail, Koza reanalyzes the failure with the entire batch of samples.

Certified Reference Material

The CRM performance range is based on the analytical performance of the Koza submissions to ALS Chemex. Koza used four CRMs prepared by and distributed by RockLabs in Auckland, New Zealand. These CRMs are SE44, HiSilK2, OxE74 and OxF65. Rocklabs CRMs are distributed with the instruction that performance ranges are based on the analytical performance of the primary laboratory during any sampling program. Koza uses ±10% initially, but once there is a statistically meaningful database Koza uses a performance range of ±2 standard deviation as described by RockLabs.

SRK reviewed Koza's internal report on the mineral resource estimate that discussed QA/QC data as well as the QA/QC database. There were 322 CRM submissions. Of these 307 CRM analyses were available as raw data. The Koza report provided graphs of all of the data. Table 2.5.1.2 lists the results of the data.

Table 2.5.1.2: Results of Au CRM Analyses at Mollakara

Standard	Number of Samples	Expected (ppm)		Observed (ppm)		% of Expected	Number Failures at ±2 Std Dev	% Failure Rate at ±2 Std Dev
		Mean	Std Dev	Mean	Std Dev			
OxF65	74	0.805	0.034	*0.797	*0.029	99.0	6	8.1
OxE74	47	0.615	0.017	0.604	0.019	98.2	10	21.2
HiSilK2	48	3.474	0.087	3.429	0.091	98.7	3	6.2
SE44	153	0.606	0.017	0.605	0.019	99.8	4	2.6
Total	**322**						**23**	**7.1**

*Does not include all analytical results for OxF65.

SRK did not have all of the raw data for OxF65, but the available data and the graphs provided in the Koza report show that the CRM reported lower grades than the overall mean with the last 22 samples reporting very high grades. Of those 22 samples, six were outside the upper limit of two standard deviations, which is two times 0.034 or ±0.07 of the mean. Other low grade CRMs also reported low overall with OxE74 showing a similar trend as OxF65. However, OxE74 started below the mean for the standard, but reported closer to the mean over time. There were nine low failures and one high failure for OxE74.

The CRM SE44 performed closest to the mean and had better performance than the other low grade CRMs, but SE44 showed a decreasing trend relative to the mean over time, which should trigger a discussion with the laboratory. Regardless, the performance of SE44 demonstrates that the laboratory is providing accurate results and that the performance observed in OxF65 and OxE74 may be the result of problems with these two CRMs and not a problem with analysis at the lab. Koza investigated the failures and have replaced OxF65 and OxE74 with SE44 based on analytical performance.

The high grade CRM, HiSilK2, also reported low grades relative to the mean. The results of the analyses indicate that the results become lower over time with three low failures. This trend is similar to that observed in SE44 and should be investigated. The majority of analyses for HiSilK2 performed within ±2 standard deviations demonstrating that the laboratory is accurate in the higher grade range.

With the exception of the six failures in OxF65, all of the standards performed within ±10% of the mean. Those that had observed performance issues were replaced with CRM SE44. The results of SE44 demonstrated that the laboratory is accurate in the lower grade range. Both SE44 and HiSilK2 had a trend that showed a lowering of the analytical results over time. This should be monitored during future drilling programs and if it continues Koza should discuss this with the laboratory. SRK also recommends that Koza monitor silver in its CRMs since a silver resource estimate is reported for Mollakara. SRK is of the opinion that the laboratory is providing accurate data and supports resource estimation.

Blanks

Prior to 2012, Koza used pulp blanks, but beginning in 2012 switched to preparation blanks. This is an appropriate blank to use to monitor cross contamination. Industry standard allows five times the detection limit as a performance range, which would be 0.025 g/t Au. Koza considers a blank a failure if it is two times the detection limit of 0.005 g/t Au, which is a more conservative approach. There were no blank failures during the drilling program at Mollakara.

Core Duplicates

Core duplicates are used early in drilling programs to identify nugget effect, determine grind size for analysis and to determine adequate sample submission size. These types of samples are generally used for a short period of time, then discontinued and replaced by preparation and pulp duplicates.

There were 178 core duplicates submitted to ALS Chemex during the course of the drilling program. Of those all were analyzed for gold and 170 were analyzed for silver. It is industry practice to use ±30% to identify core duplicate failures. Koza uses ±20% to identify duplicate failures, which is more conservative. Table 2.5.1.3 presents the results for core duplicate analyses for gold.

Table 2.5.1.3: Summary of Duplicate Gold Analysis at Mollakara

Criteria	Number of Samples	Original>Dup	Dup>Original	Original = Dup	Within +/-20%	Within +/-30%
All samples	178	90	84	4	131	151
		51%	47%	2%	74%	85%

The duplicates did not show a strong high or low bias but were distributed relatively equally on both sides of the x = y line. The results suggest that there may be a slight nugget affect and that there is reasonable repeatability for the samples. SRK notes, that using ±30%, there were 20 duplicate failures in the 94 sample pairs above the cutoff grade of 0.22 g/t Au for resources at Mollakara. The majority of these failures were at lower grades suggesting that the failures may be related to analytical precision near the cutoff grade as opposed to an actual nugget or sample size problem. However, analytical precision is better tested with pulp duplicates and core duplicates may not be sufficiently similar to provide an adequate assessment of precision.

Silver duplicates had a 22% failure rate or 38 failures in 170 samples. The majority of these failures were at low grades between the detection limit and 0.35 g/t Ag. As with gold, this suggests that the failures are related to the analytical precision near the detection limit.

Based on the performance of the core duplicates, SRK recommends that during subsequent drilling programs, Koza discontinue the use of core duplicates unless there is a significant change in mineralization such as texture and grain size of key minerals and/or a new area of the deposit is drilled that has significantly different geology and mineralization style. SRK also recommends that Koza incorporate preparation and pulp duplicates into the QA/QC program as well as submitting a subset of the pulps to a second laboratory to check analysis at ALS Chemex.

When a failure occurs, Koza assesses the failure and decides on a course of action. If it is only one failure, Koza reanalyzes five samples before and after the failure. However, in the case of multiple failures, Koza may reassay the entire batch. These actions are industry practice.

SRK also recommends that Koza consider the following performance gates for CRMs:

- If one analysis is outside of ±2 standard deviations it is a warning;
- Two or more consecutive analyses outside of ±2 standard deviations is a failure;
- If an analysis is outside ±3 standard deviations it is a failure if ±3 standard deviations does not exceed ±10% of the mean; and
- If the ±3 standard deviations exceed ±10% of the mean, then ±5 to ±10% should be used.

Ore Research & Exploration (OREAS), who manufactures CRMs, recommends using these performance gates and has started printing this information on CRM certificates as part of a guide for use of the CRM. ALS Global uses ±3 standard deviations during analysis as a performance gate for internal CRMs (ALS Global, 2012). Koza is using a more restrictive performance gate that may result in unnecessary failures.

The results of the QA/QC samples at the Mollakara project indicate that the laboratory performance is acceptable and that the database can be used for resource estimation.

2.6 Mineral Resources

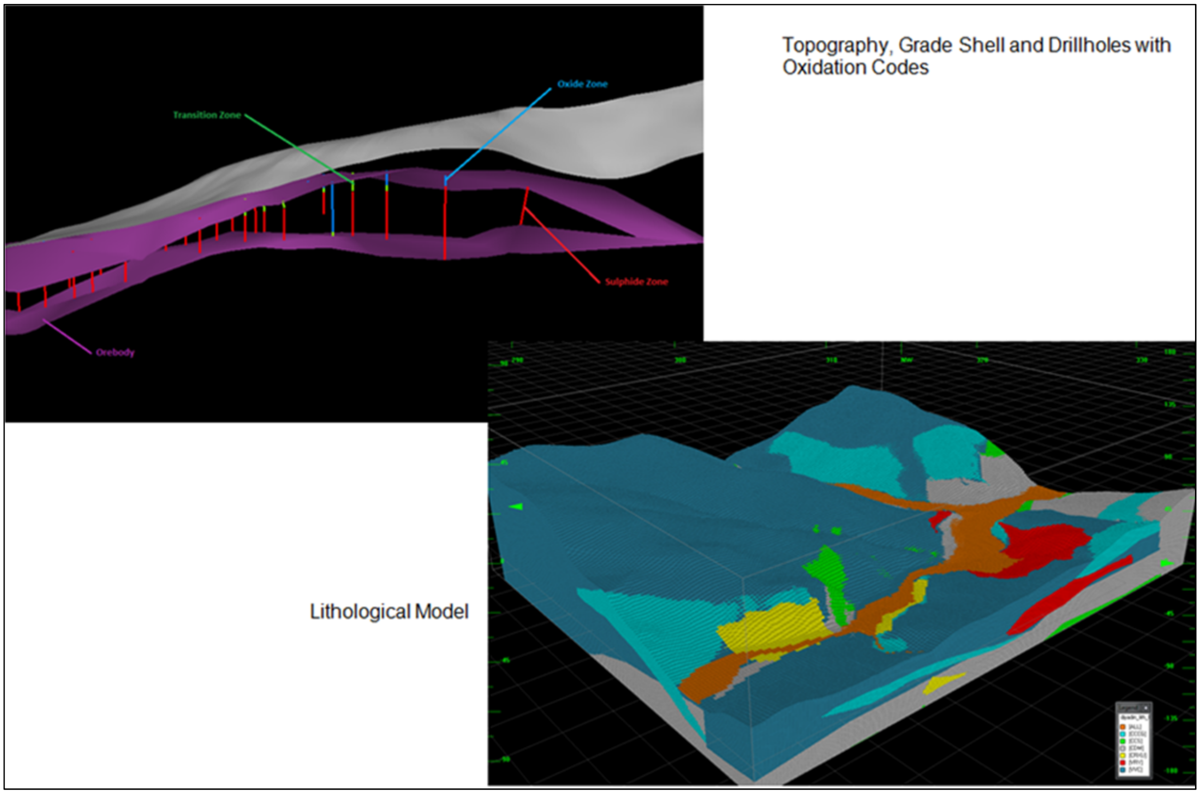
The Mollakara Mineral Resources were estimated in 2011 by Koza (Koza, 2011).

2.6.1 Geological Modeling

The Mollakara resource database consists of 27 RC drillholes and 125 core holes for a total of 18,600.4 m of drilling in 152 holes. The drilling is on a grid pattern of about 50 by 50 m in the center of the deposit, expanding outward to about to over 200 m at the outermost drillholes.

Core recovery ranges from 20 to 100% with an average of 96%. Koza prepared a scatter plot of gold versus core recovery. The plot indicates that the samples with recovery less than 50% showed a high bias and those samples (14 total) were removed from the resource database.

Koza constructed wireframe surfaces for oxide, transition and sulfide zones based on the drillhole logging. The oxide zone varies from 0 to 45 m in thickness and the sulfide varies from 20 to 120 m in thickness. The transition zone is a thin semi-continuous layer up to 15 m in thickness. Koza also constructed lithological and alterations models for Mollakara. A grade shell was constructed at 0.1 g/t Au. Figure 2.6.1.1 illustrates the grade shell and lithological models.



Source: Koza, 2011

Figure 2.6.1.1: Mollakara Geological Model

The grade shell covers an area of about 1,700 m northeast to southwest and 1,300 m northeast to southwest. The thickness is greatest in the center of the deposit (100 m) and thins to less than 10 m at the edges of the drilling. The more closely drilled area (roughly 60 to 70 m spacing) is about 750 m by 450 m in the same directions. The grade shell has been constructed without use of the lithologic model or the faults.

A total of 16,307 drillhole intervals have been analyzed for gold and silver. There are 10,413 samples within the grade shell. Statistics of the assays within the grade shell are given in Table 2.6.1.1.

Table 2.6.1.1: Statistics of Assays within the Grade Shell at Mollakara

Type	Variable	Number	Min	Max	Mean	S.D.	CV
Oxide	Au	2257	0.005	10.10	0.75	0.86	1.15
	Ag	1998	0.010	14.00	0.21	0.73	3.43
Transition	Au	638	0.005	8.51	0.81	0.92	1.14
	Ag	498	0.010	22.40	0.26	1.17	4.56
Sulfide	Au	7518	0.005	14.45	0.92	1.04	1.13
	Ag	7294	0.005	235.00	0.26	2.99	11.67
Total	Au	10413	0.005	14.45	0.88	1.00	1.14
	Ag	9790	0.005	235.00	0.25	2.61	10.58

The gold values in RC and core holes were plotted on a QQ graph to evaluate differences in gold distribution between the two drilling types. The plot shows that up to about 1 g/t that the populations are very similar. Over 1 g/t, there is a slight high bias to the RC samples. Koza concluded that there was no difference between the two populations and that both were suitable for use in resource estimation. SRK notes that there is a slight bias at the upper range, but that there are relatively few RC samples compared to core samples and agrees that it is correct to use both types of drilling.

2.6.2 Capping and Compositing

The sample lengths were plotted on a histogram to evaluate the compositing length; the vast majority of samples are 1 m in length, but there is a second spike at 2 m. Koza composited the samples on 2 m lengths from the top of the drillhole with breaks where the drillhole enters and exits the grade shell.

The gold and silver composites were reviewed using a quantile analysis to determine the need for capping. The gold values in the oxide zone were capped at 4.3 g/t and the silver grades were capped at 4 g/t. In the transition zone, gold was capped at 4.1 g/t and silver was capped at 4 g/t; in the sulfide zone, gold was capped at 5.5 g/t and silver was capped at 6 g/t. A total of 20 gold samples and 14 silver samples were capped. Table 2.6.2.1 presents the statistics for the capped composites. The Coefficient of Variation (CV) has been brought below 1 for Au through capping and compositing, but is still relatively high at over 2 for silver.

Table 2.6.2.1: Statistics of Capped Composites at Mollakara

Type	Variable	Number	Min	Max	Mean	S.D.	CV
Oxide	Au	1157	0.02	4.30	0.75	0.73	0.98
	Ag	1023	0.01	4.00	0.20	0.44	2.22
Transition	Au	321	0.01	4.10	0.80	0.81	1.00
	Ag	250	0.01	4.00	0.23	0.47	2.06
Sulfide	Au	3771	0.01	5.50	0.91	0.88	0.97
	Ag	3655	0.01	6.00	0.20	0.48	2.46
Sulfide	Au	5249	0.01	5.50	0.87	0.85	0.98
	Ag	4928	0.01	6.00	0.20	0.47	2.39

2.6.3 Variography

Koza conducted variography studies on samples from the three oxidation types. There were too few transition composites to obtain reliable variograms. Table 2.6.3.1 gives the variogram parameters for oxide and sulfide composites.

Table 2.6.3.1: Mollakara Variogram Parameters

Zone	Axis	Orientation	Nugget	Sill 1	Sill2	Sill3	Range1 (m)	Range2 (m)	Range3 (m)
Oxide Au	Major	00°,000°	0.050	0.384	0.052	0.047	9	47	125
	Semi-major	00°,000°					9	47	125
	Minor	00°,000°					9	47	125
Oxide Ag	Major	00°,000°	0.100	0.074	0.067	0.044	10	45	105
	Semi-major	00°,000°					10	45	105
	Minor	00°,000°					10	45	105
Sulfide Au	Major	10°,270°	0.090	0.360	0.328		46	83	
	Semi-major	08°,180°					51	127	
	Minor	00°,190°					32	10	
Sulfide Ag	Major	00°,000°	0.032	0.115	0.083		10	82	
	Semi-major	00°,000°					10	82	
	Minor	00°,000°					10	82	

2.6.4 Specific Gravity

Specific gravity was measured on 371 core samples from 126 drillholes. The samples were grouped by rock type and by oxidation state. The specific gravity determinations were made using the Archimedes principle where the core was covered with wax and the samples were weighed in water and in air. Table 2.6.4.1 shows the specific gravity values assigned to the block model by lithology and oxidation type. The specific gravity is on a dry basis.

Table 2.6.4.1: Mollakara Specific Gravity

Code	Lithology	SG Mineralized			SG Waste
		OX	TRAN	SULF	
CCCG	Carbonate clast conglomerate	2.45			2.45
CCS	Upper Calc-Schist, Lower Calc-Schist	2.43	2.57	2.58	2.52
CDM	Upper medium-bedded dolomite, Lower medium-bedded dolomite	2.58	2.69		2.72
CRXU	Undivided carbonate rocks	2.60			2.51
VVC	Volcanic epiclastics, Lower tuff, Welded ash flow tuff, Carbonaceous silty landslide, Debris Flow	2.21	2.2		2.09

Source: Koza 2011

2.6.5 Grade Estimation

A block model was constructed with a cell size of 10 m x 10 m x 5 m and with sub-blocking to 2.5 m x 2.5 m by 1.25 m at the topography and oxidation surfaces and at the grade shell boundary. The block size is about 20% of the drill spacing. Variables in the block model include gold, silver, arsenic, oxidation state, lithology, density, number of drillholes and samples used in the estimation and the closest distance between the block center and the composites used in estimation. The estimation was conducted separately for each of the oxidation types, using only composites with the corresponding code. Only blocks inside the grade shell were estimated and all composites had to be within the grade shell.

The estimation was done with four passes, using ordinary kriging (OK) for oxide and sulfide and inverse distance squared (ID2) for transition. The estimation was carried out at the parent cell size and Datamine's dynamic search option was used. The parameters used in the four estimation passes are shown below:

- First: search two-thirds of the maximum variogram range, minimum of 12 and maximum of 25 composites in the oxide and sulfide zones and a minimum of 10 and a maximum of 20 in the transition zone. An octant search was used, requiring a minimum of 3 octants with a maximum of 4 samples per octant;
- Second: search 1.5 times the maximum variogram range with same minimum and maximum number of composites as the first pass;
- Third: search 3 times the maximum variogram range with the same minimum and maximum number of composites as the first pass; and
- Fourth: search 5 times the maximum variogram range with a minimum of 2 and a maximum of 15 samples with a maximum of 3 per drillhole.

2.6.6 Block Model Validation

Koza validated the block model using three methods including a visual comparison of block and drillhole grades on cross-sections and plans, comparison of average grades of the composites to the block grades and by conducting a Nearest Neighbor estimation.

Table 2.6.6.1 presents a comparison of the composite grades to the estimated grades and the nearest neighbor estimation.

Table 2.6.6.1: Comparison of Composites and Estimated Grades at Mollakara

Zone	Variable	Composites	OK	ID2	NN
Oxide	Au	0.75	0.585	0.587	0.571
	Ag	0.20	0.147	0.140	0.138
Transition	Au	0.80		0.645	0.659
	Ag	0.23		0.151	0.160
Sulfide	Au	0.91	0.801	0.802	0.811
	Ag	0.20	0.157	0.153	0.165

The gold and silver grades produced by the three estimation methods are very close to each other and are all less than the composite grades used in the estimation.

2.6.7 Mineral Resource Classification

The blocks which had been estimated in the first pass with at least 4 drillholes and an average distance of less than 25 m to the block centroid were plotted on a map and a polygon drawn around those blocks was used to classify the Measured Resource. Similarly, blocks estimated in the first pass with at least 3 drillholes and an average distance of less than 60 m to the block centroid were identified and a polygon drawn around them to classify Indicated resources. The remainder of the blocks were classified as Inferred. SRK finds this to be a very good method of classification as it is based primarily on sample spacing and eliminates irregularities that can be seen when only the estimation pass is used in the classification. Figure 2.6.7.1 shows the classification polygons with the drillholes and grade shell.

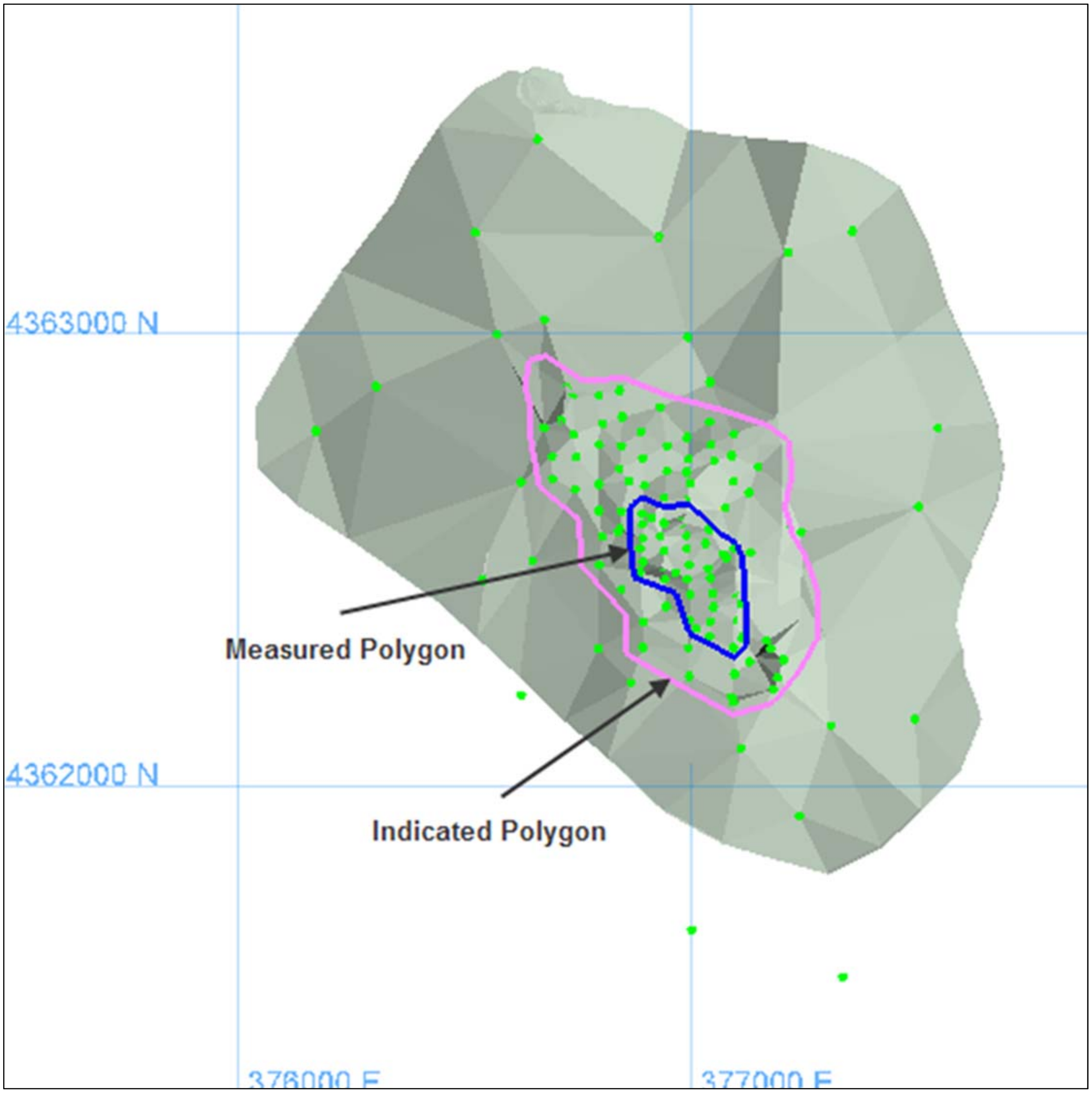


Figure 2.6.7.1: Mollakara Grade Shell, Drillholes and Measured and Indicated Polygons

2.6.8 Mineral Resource Statement

The cutoff grades were calculated from the parameters in Table 2.6.8.1. The one year rolling average gold price is US$1,266; the two year average is US$1,339; and the three year average is US$1,449. The cutoff grade is based on the assumption that the oxide and transition will be mined by open pit methods and will be processed by heap leaching; the assumption for the sulfide is that it will be mined by open pit methods and that a mill will be constructed on site for processing the sulfide material. There have been no metallurgical tests on the sulfide material to support the cutoff and no studies to indicate the viability of building a mill on the Mollakara site. SRK recommends that Koza conduct metallurgical test work to identify the process route for the sulfide material and justify the use of this cutoff grade and inclusion in the resource.

Table 2.6.8.1: Mollakara Cutoff Grade Parameters

Prices and Costs	Units	Oxide	Transition	Sulfide
Gold Price	US$/oz	1,450	1,450	1,450
Gold Recovery	%	0.65	0.36	0.90
Gold Refining	US$/oz	3.44	3.44	3.44
Government Right	%	1	1	1
Process Cost	US$/t	4.80	4.80	15.00
Mining Cost	US$/t	0.00	0.00	0.00
G&A Cost	US$/t	1.00	1.00	1.00
Ore Rehandling	US$/t	2.00	2.00	0.00
Calculated Cutoff grade	g/t	0.26	0.48	0.39
Final Cutoff grade	g/t	0.26	0.48	0.39

Source: Koza, 2014

It is SRK's policy to report resources within a pit optimization shell to meet JORC requirements that resources be potentially mineable. The resources at Mollakara fall entirely within the pit shell run with the parameters in Table 2.6.8.1.

A portion of the Mollakara deposit underlies the Murat River and one of its tributaries. Koza has assumed that the rivers can be diverted, and that the permitting can be obtained for the diversion, to allow for open pit mining of the entire resource.

The resources are listed by oxidation type in Table 2.6.8.2.

Table 2.6.8.2: Mollakara Mineral Resources, including Ore Reserves, at December 31, 2014

Classification	kt	g/t Au	g/t Ag	koz Au	koz Ag
Oxide					
Measured	2,942	0.80	0.2	76	20
Indicated	9,414	0.73	0.2	222	58
Measured and Indicated	12,356	0.75	0.2	298	78
Inferred	7,426	0.47	0.1	112	23
Transition					
Measured	570	1.26	0.4	23	7
Indicated	2,570	0.86	0.2	71	18
Measured and Indicated	3,140	0.93	0.2	94	24
Inferred	4,582	0.69	0.1	102	18
Sulfide					
Measured	9,481	1.11	0.2	338	55
Indicated	34,123	0.98	0.2	1,080	227
Measured and Indicated	43,604	1.01	0.2	1,418	282
Inferred	94,064	0.83	0.1	2,520	435
Total					
Measured	12,993	1.05	0.19	437	81
Indicated	46,107	0.93	0.20	1,373	303
Measured and Indicated	**59,100**	**0.95**	**0.20**	**1,810**	**385**
Inferred	106,072	0.80	0.14	2,733	477

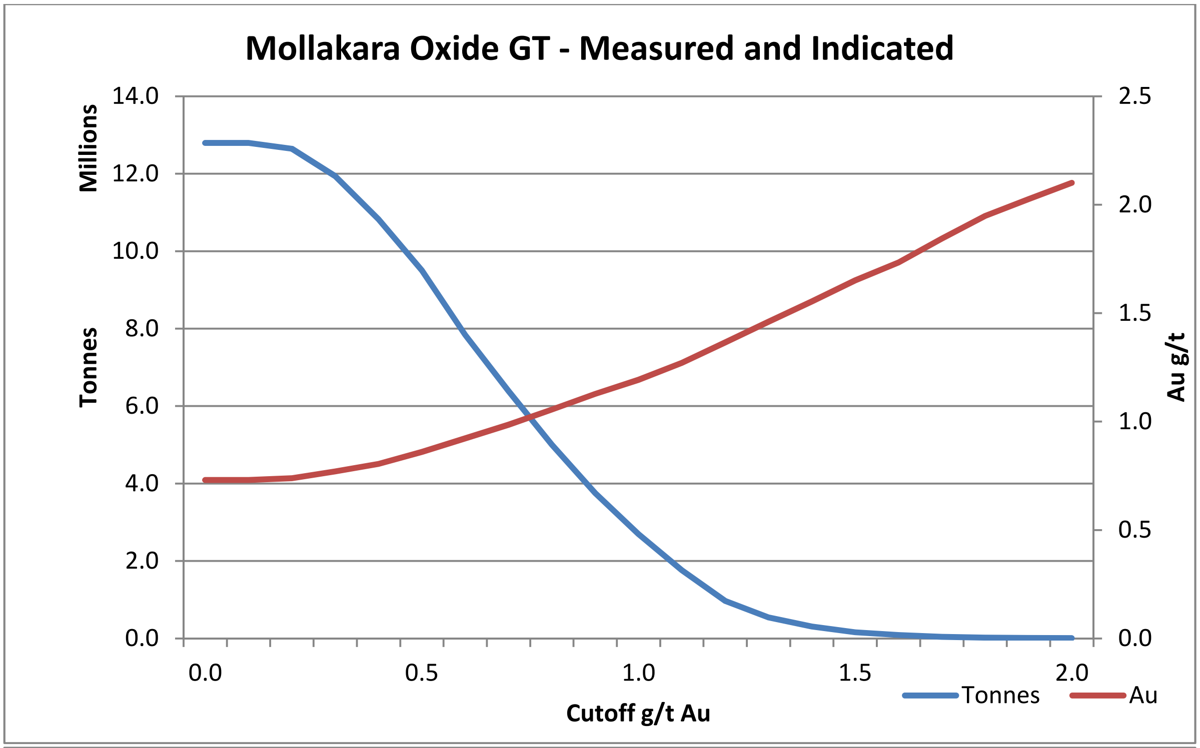
- Tonnages and grade are rounded to reflect approximation;
- Resources are stated at a cutoff grade of 0.26 g/t Au for oxide, 0.48 for transition and 0.39 g/t Au for sulfide;
- Open pit resources are contained within grade shells but are not constrained by a pit optimization shell; and
- Mineral Resources are reported inclusive of Mineral Reserves.

2.6.9 Mineral Resource Sensitivity

Grade tonnage curves for the Oxide, Transition and Sulfide combined Measured and Indicated resources and Inferred resources are presented in Figures 2.6.9.1, 2.6.9.2 and 2.6.9.3. Cutoff grades for the Mollakara resource at various gold prices are shown in Table 2.6.9.1.

Table 2.6.9.1: Mollakara Cutoff Grades vs. Gold Price

	Cutoff Grade		
Gold Price	Oxide	Transition	Sulfide
1600	0.24	0.43	0.35
1550	0.24	0.45	0.36
1500	0.25	0.46	0.37
1450	0.26	0.48	0.39
1400	0.27	0.49	0.40
1350	0.28	0.51	0.41
1300	0.29	0.53	0.43
1250	0.30	0.55	0.45
1200	0.24	0.43	0.35



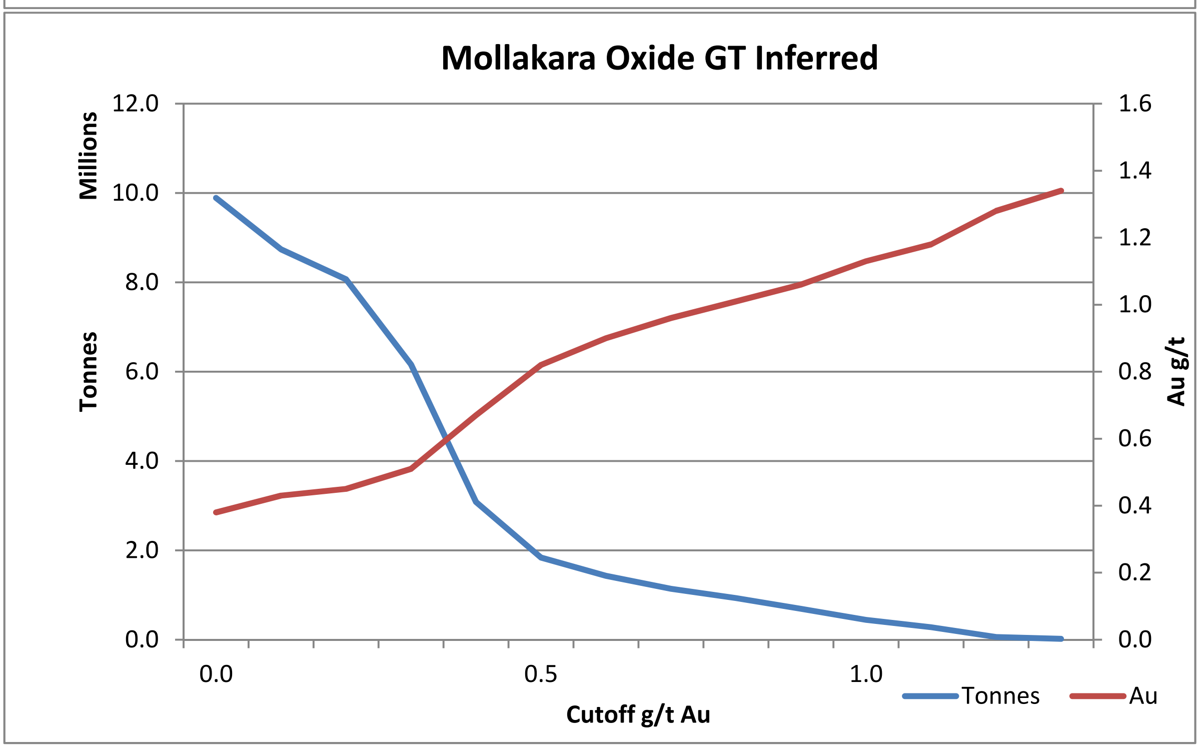
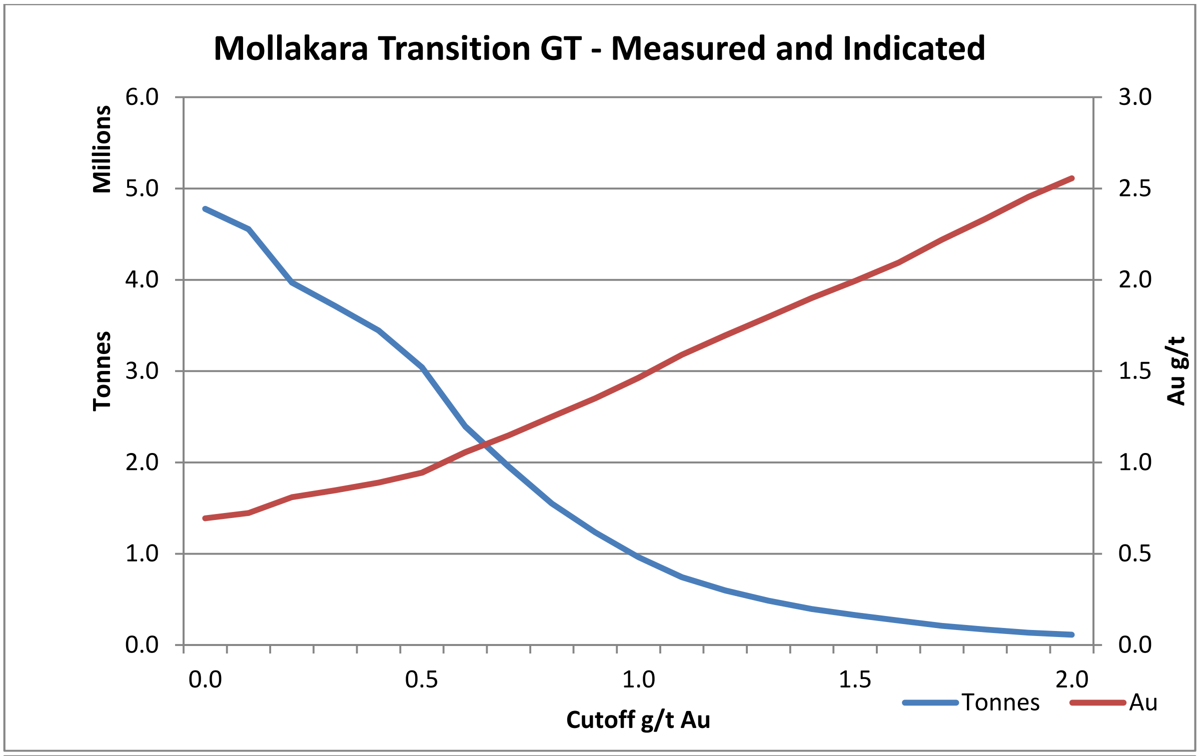


Figure 2.6.9.1: Grade Tonnage Curve Mollakara Oxide Measured, Indicated and Inferred Resources



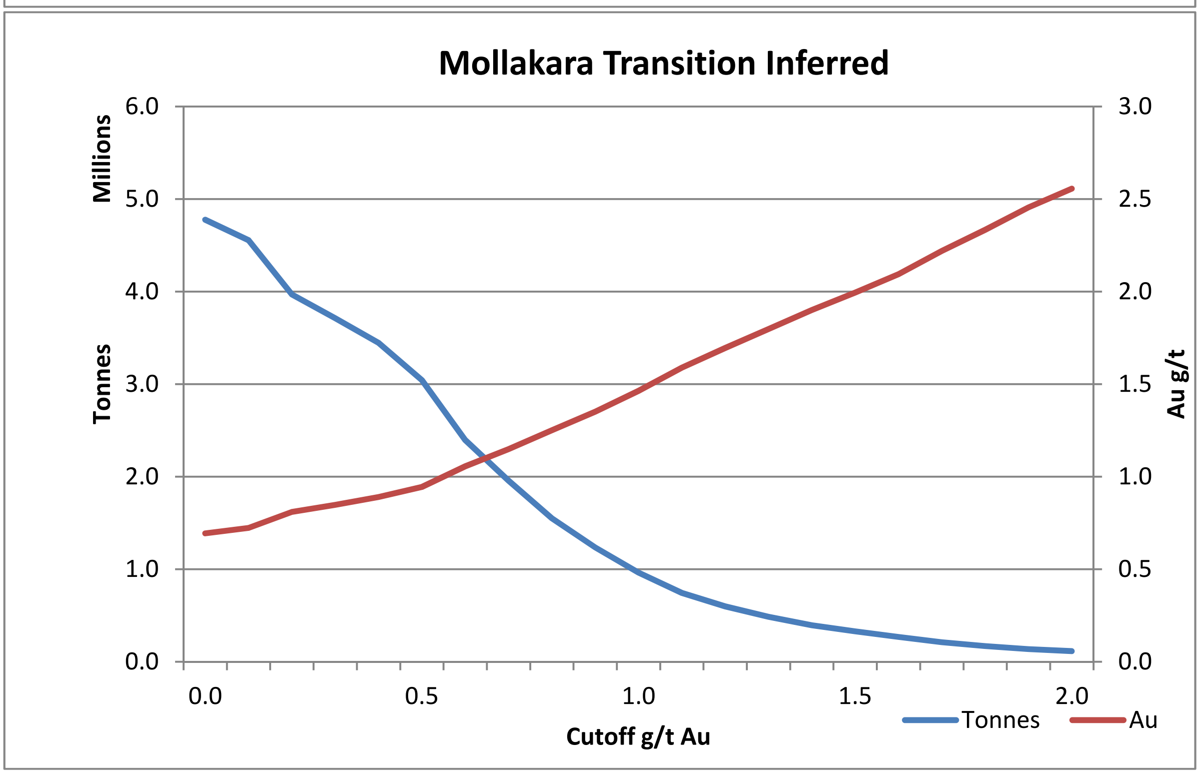
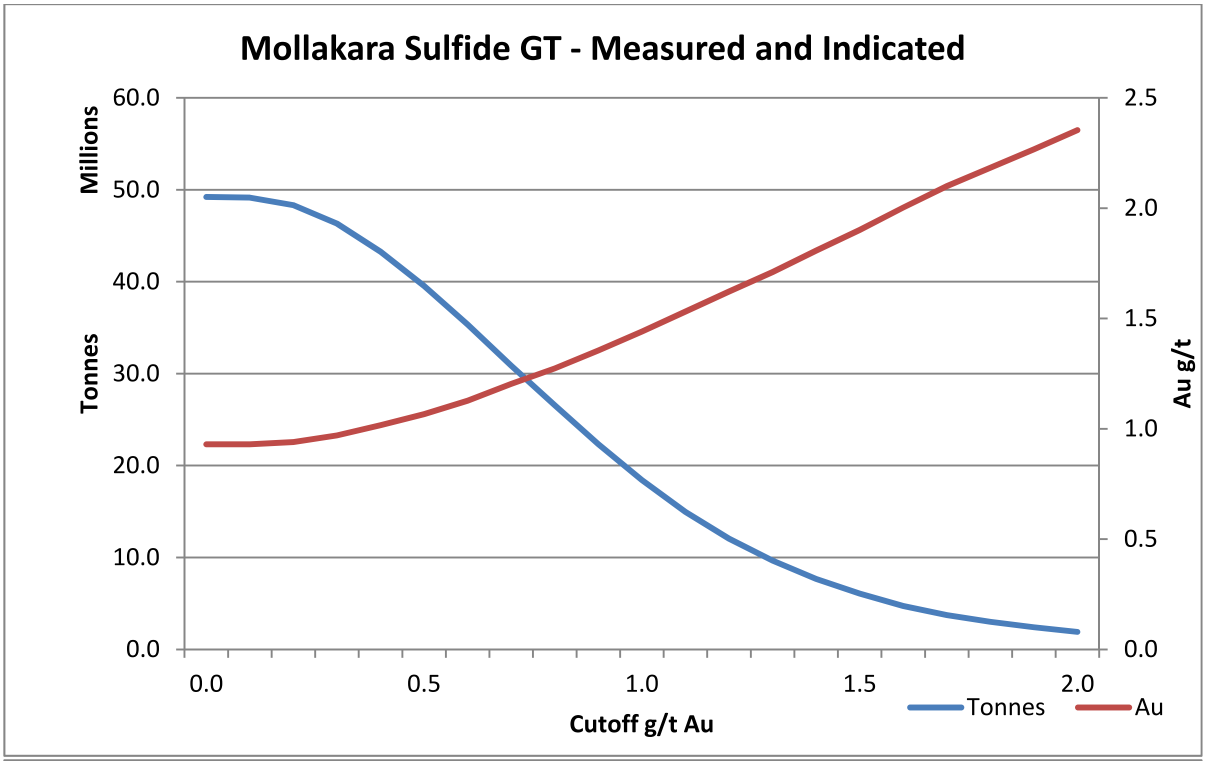


Figure 2.6.9.2: Grade Tonnage Curve Mollakara Transition Measured, Indicated and Inferred Resources



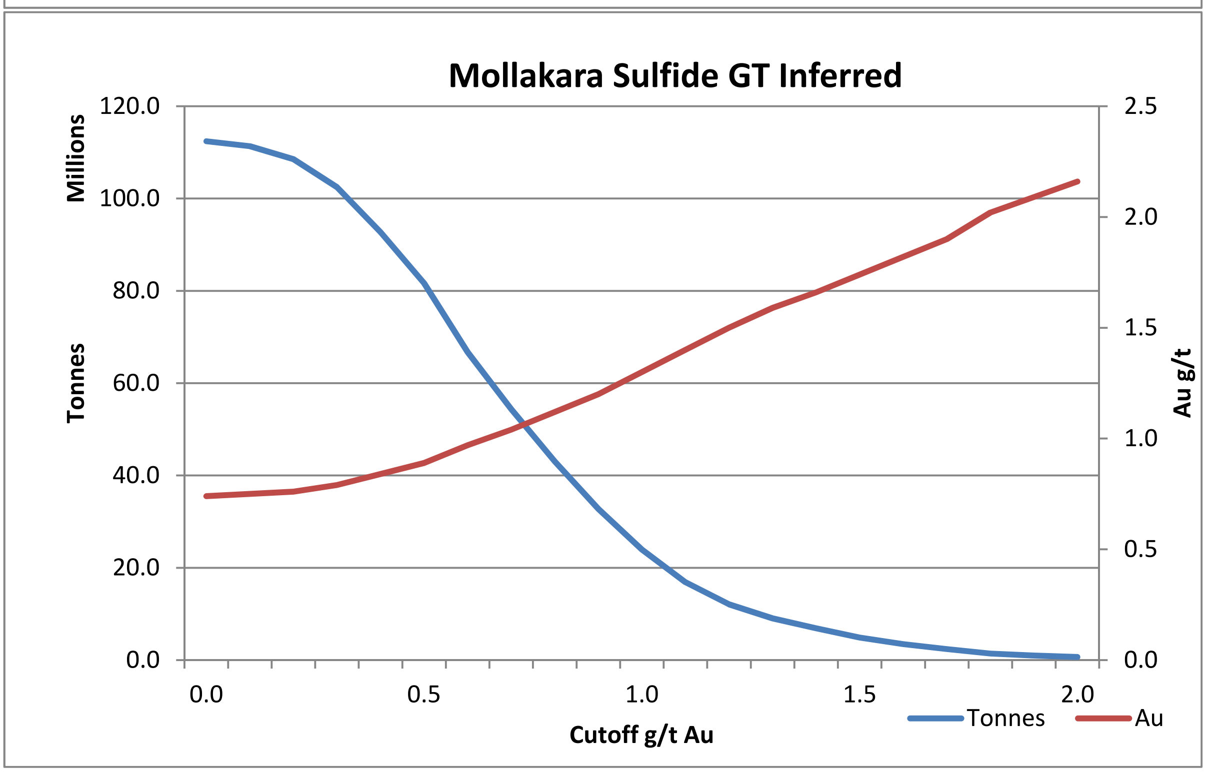


Figure 2.6.9.3: Grade Tonnage Curve Mollakara Sulfide Measured, Indicated and Inferred Resources

2.7 Ore Reserve Estimation

Mollakara is a development Project with mine production expected to begin in April 2019 and run through July 2021. Updated metallurgical, costing and geotechnical information received during 2012 escalated the Project from potentially mineable to reserve status.

LoM plans and resulting reserves are determined based on a gold price of US$1,250/oz for the open pit mine project. Reserves stated in this report are as of December 31, 2014.

The ore at Mollakara is to be extracted using open pit mining methods and heap leach gold extraction. The ore material is converted from resource to reserve based primarily on positive cash flow pit optimization results, technical economic model, mine design and geological classification of Measured and Indicated resources. The in-situ value is derived from the estimated grade and various modifying factors.

Modifying Factors

The conversion of resource to reserve entails the evaluation of modifying factors that should be considered stating a reserve. Table 2.7.1 illustrates a reserve checklist and associated commentary on the risk factors involved for the Mollakara reserve statement.

Only oxide and transition material are considered as reserve and the pit has been limited by the Murat River.

Table 2.7.1: Reserve Checklist and Associated Commentary on the Risk Factors

Unit	Data Evaluated	Data Not Evaluated	Not Applicable	Notes
Mining				
Mining Width	X			Small mining trucks – 10m x 10m x 5m
Open Pit and/or Underground	X			Open pit
Density and Bulk Handling	X			Contractor mining
Dilution	X			SMU 10 x10 x 5
Mine Recovery	X			Full mine recovery assumed
Waste Rock	X			Waste dump strategy in place and sufficient volume
Grade Control	X			Blast holes on 4m x 4.5m pattern
Processing				
Representative Sample	X			Heap leach studies by McClelland and Kaymaz Lab
Deleterious Elements	X			Clay, possible percolation issues
Process Selection	X			Heap leach pad
Geotechnical/Hydrological				
Slope Stability (Open Pit)	X			Slope stability study with groundwater treated as saturated
Area Hydrology	X			Dry climate – River used to limit pit extents. Perched aquifer
Seismic Risk	X			Geotechnical study performed – evaluated in design for slopes and heap pad
Environmental				
Baseline Studies	X			EIA
Tailing Management			X	No tailings at site
Waste Rock Management	X			Waste area located
Acid Rock Drainage Issues		X		Oxide mining
Closure and Reclamation Plan		X		Project still developing EIA
Permitting Schedule		X		Ongoing
Legal Elements or Factors				
Security of Tenure	X			Assume no limiting factor to Mining
Ownership Rights and Interests	X			Assume no limiting factor to Mining; Resource defined
Environmental Liability	X			Assume no limiting factor to Mining; Resource defined
Political Risk (e.g., land claims, sovereign risk)	X			Close to Iran border, political unrest in region. Koza and government relations are poor.
Negotiated Fiscal Regime	X			Assume no limiting factor to Mining; Resource defined
General Costs and Revenue Elements or Factors				
General and Administrative Costs	X			
Commodity Price Forecasts	X			
Royalty Commitments	X			
Taxes	X			
Corporative Investment Criteria	X			
Social Issues				
Sustainable Development Strategy	X			Koza Environmental/Social Plan – First mine in region
Impact Assessment and Mitigation		X		Koza Environmental/Social Plan

Unit	Data Evaluated	Data Not Evaluated	Not Applicable	Notes
Negotiated Cost/Benefit Agreement		X		Assume no limiting factor to mining
Cultural and Social Influences	X			Koza Environmental/Social Plan. Concerns for local stakeholders.

Source: SRK, 2014

Through the process of pit optimization, the economics associated with the project are used to determine the economic viability of the mine design and production schedule. It should be noted that the pit design for 2014 is the same as 2012 because the pit is not economically constrained but rather by the location of the river and transition/sulfide boundary. For 2014, the cutoff grade was changed based on the US$1,250 gold price and inclusion of the transportation cost.

Table 2.7.2 details the cost breakdown for pit optimization. Mining costs have been estimated using Koza's experience with Turkish contractors and initial quotations. Processing cost is based on reagent consumptions and associated infrastructure while rehabilitation, grade control, administration and selling cost are from prior operational experience throughout Koza sites.

Table 2.7.2: Mollakara Pit Optimization Inputs (as of December 30, 2014)

Parameter	Unit	Oxide	Transition
Mining Cost	US$/t material	1.49	1.49
Rehabilitation Cost	US$/t waste	0.20	0.20
Heap Leach Cost	US$t/ore	4.98	4.98
Selling Cost	US$/oz	3.44	3.44
Grade Control	US$t/ore	0.5	0.5
Administration	US$t/ore	1	1
Gold Price	US$/oz	1,250	1,250
Silver Price	US$/oz	20	20
Gold Recovery	%	65	36
Silver Recovery	%	10	10
Cutoff grade	g/t Au	0.33	0.59

Source: Koza, 2014

2.7.1 Reserve Classification

Table 2.7.1.1 details the mineable reserves for Mollakara.

Table 2.7.1.1: Mollakara Reserves, at December 31, 2014

Category	Kt	g/t Au	g/t Ag	Contained oz Au	Contained oz Ag
Proven Reserve	3,529	0.87	0.2	99	27
Probable Reserve	11,387	0.75	0.2	275	71
Total Proven and Probable Reserve	14,916	0.78	0.2	374	98

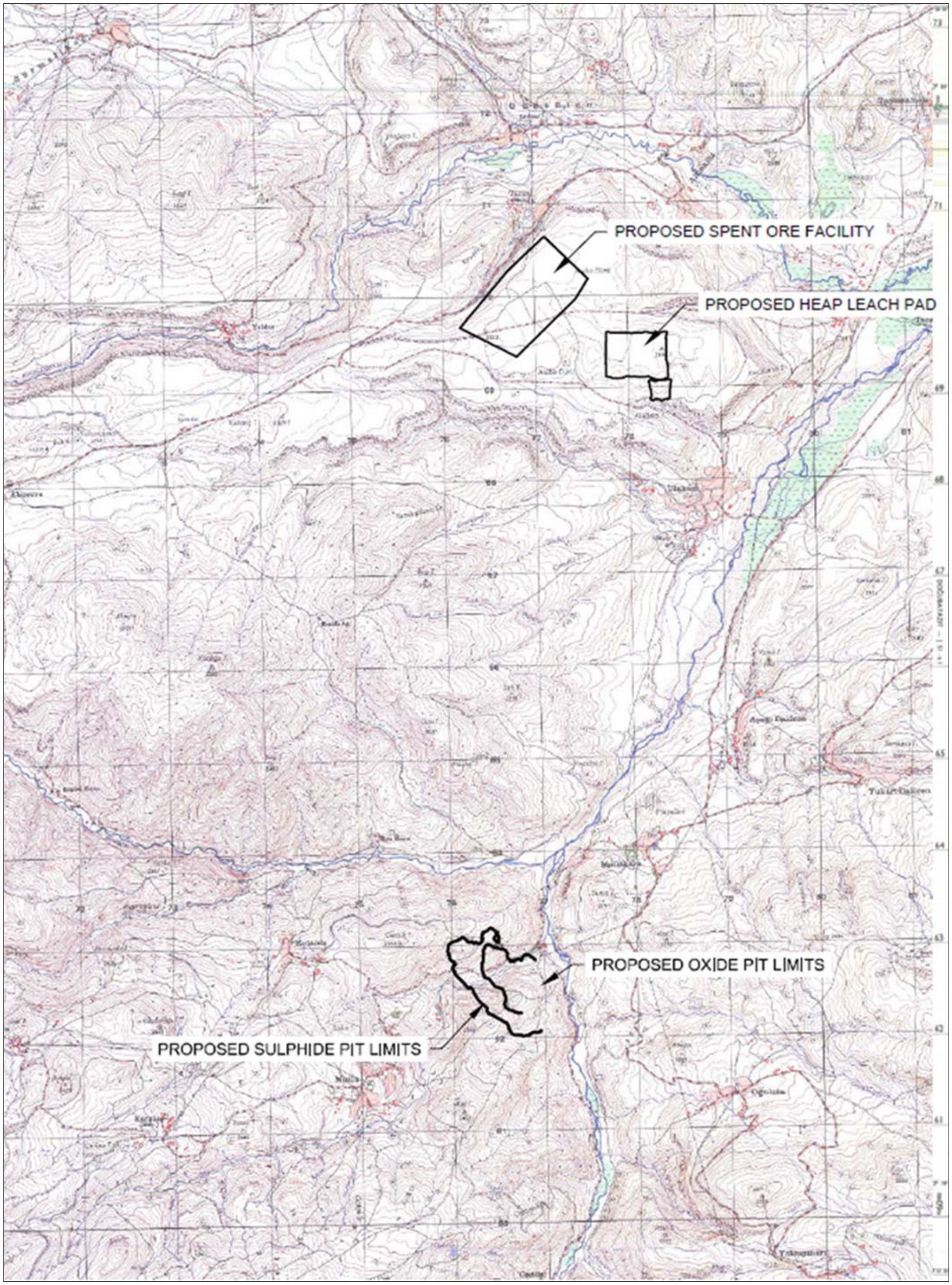
Metal price: US$1,250/oz-Au, US$20/oz-Ag, Au Recovery 65%, Ag Recovery 10%, Oxide Au Cutoff grade 0.33 g/t. Transition Au Cutoff grade 0.59 g/t.

Sulfide resources are not included in the Mollakara reserves.

2.8 Mining Engineering

SRK conducted a site visit of the Mollakara project in November 2010. The Mollakara orebody is located on the side of a valley surrounded by hilly terrain in eastern Turkey. The orebody is intersected by a small/medium sized river and comprises oxide and transition material which is suitable for heap leaching and sulfide material which continues under the river. Due to the limited space for heap leach pads, waste dump locations, and the environmental concerns of mining near a major watershed, only the oxide and transition material is being studied at this time.

Mine production is expected to begin in 2019 based on the assumption that all necessary permits and detailed design studies will have been carried out. Due to the difficult mining conditions faced during the winter months, Koza are planning on mining between March and November each year with approximately 7 Mt of leach material stockpiled at the heap leach pad at maximum production. As of 2012, the proposed leach pad location has been sourced and is illustrated in Figure 2.8.1.



Source: SRK 2012

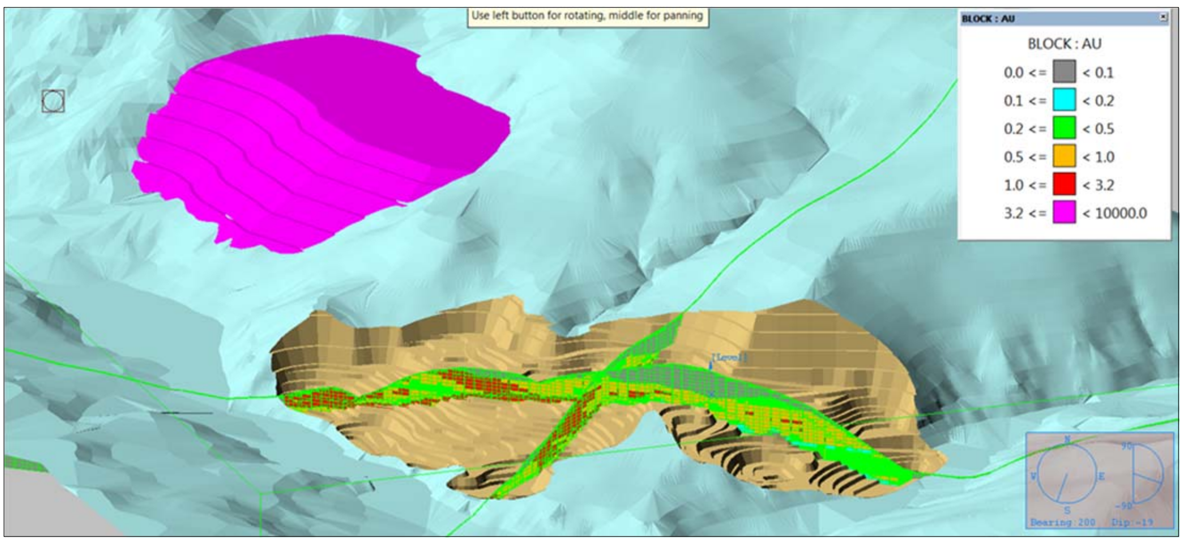
Figure 2.8.1: Mollakara Heap Leach Pad Location

Waste is planned to be managed south of the open pit and the current design defines space for 7.2 Mm^3 which is enough for disposal of waste associated with the existing reserves.

An additional constraint placed on the oxide and transition pit is the location of the Murat River. Only a small buffer has been left between the river level (which corresponds to groundwater table) and the pit rim. During detailed design it will be necessary to determine if any hydraulic conductivity exists between the river and pit sump.

Mining will be carried out by contractors and run in the same fashion as all other Koza sites using 40 t highway trucks and associated loaders. The one exception will be grade control that will be dependent on sampling of blast hole chips, as the channel sample method used at other Koza operations is not applicable to disseminated orebodies.

Figure 2.8.2 Illustrates the open pit design, waste dump locations and block model sections of the Mollakara reserves.



Source: SRK 2013

Figure 2.8.2: Mollakara Pit and Dump

Approximately 12 Mt will be excavated in the first year followed by 10 Mt in the second and 5 Mt in the third year of operations. Until reaching the fourth bench, blasting will be initiated and excavated on 5 m benches. Berms will be placed every 20 m, but excavators will mine two 2.5 m flitches for each 5 m of blasted bench.

The general blast hole layout will comprise of 5.2 m deep holes placed on a 4 m burden and 4.5 m spacing. After drilling the blast holes, samples will be taken and sent to the laboratory by Koza personnel.

<u>**Geotechnical**</u>

No updates to the geotechnical analysis occurred in 2014.

The topography around the Mollakara pit infers natural slumping and therefore geotechnical analysis has been a high priority as exploration began. SRK Turkey defined the groundwater table to be at elevation 2055 m RL, just above the pit bottom of 2040 m RL and river level.

Using a reserve level pit design, Koza staff used a geotechnical software, Slide, to test three scenarios on section defined in Figure 2.8.3:

- Water Level at +2055 m RL;
- Saturated groundwater conditions with maximum pore pressure (h_u*=1.0); and
- Saturated groundwater conditions with 40% head loss due to seepage (h_u*=0.6).

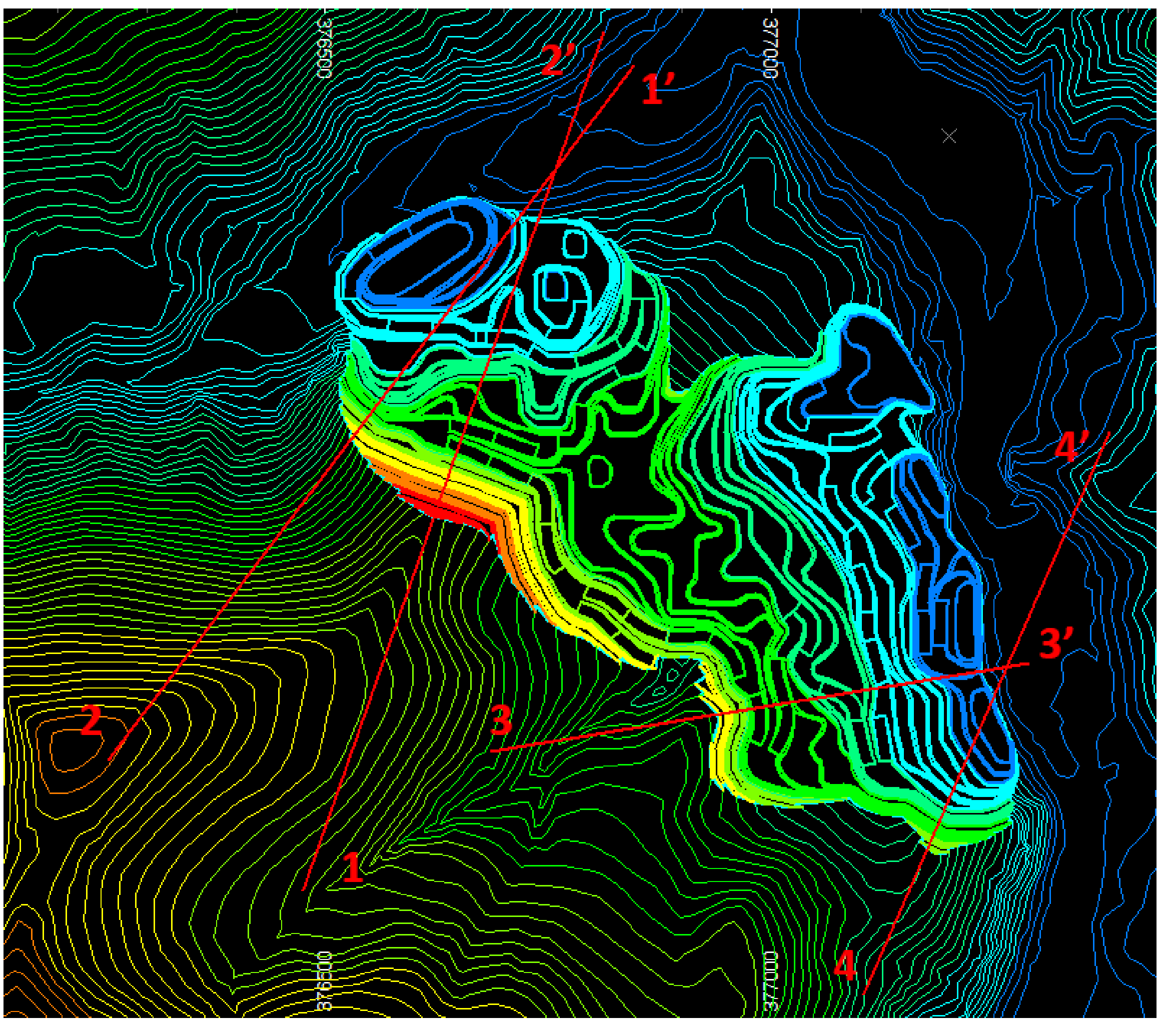
Table 2.8.1 details the base parameters used for the geotechnical analysis by lithological formation modeled.

Table 2.8.1: Mollakara Strength Parameters

Lithologic Formation	σci (MPa)	GSI	mi	σ3max	Disturbance Factor	c (kPa)	Φ (°)
VVC	59.11	40	7.95	1.86	1.0	283	24.36
CCS	63.10	40	10.72	1.90	1.0	323	27.12

Source: Koza 2012

Figure 2.8.3 illustrates the sections analyzed and Table 2.8.1 details the proposed ramp angles.



Source: Koza, 2012

Figure 2.8.3: Mollakara Geotechnical Sections

The details of the section lines shown in Table 2.8.2 and include ramps and boundary between oxide and sulfide material.

Table 2.8.2: Proposed Ramp Angles

Cross-Section	Slope Height (m)	Slope Angle (°)
Section 1	107	49
Section 2	76	47
Section 3	125	21
Section 4	90	41

Source: Koza, 2012

As a result of the analysis, Table 2.8.3 details the resultant factor of safety analysis and design inputs based on the water table, saturated and semi-saturated slope analysis.

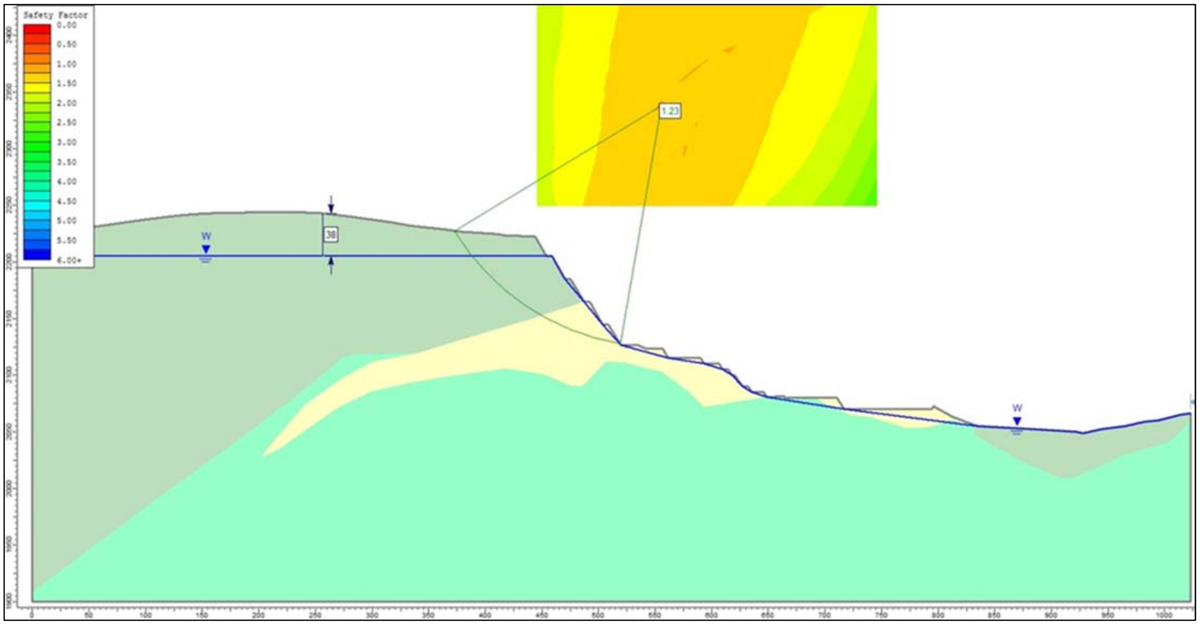
Table 2.8.3: Resultant Factor of Safety Analysis and Design Inputs

	Dry Slope	Saturated Slope (h_u=1.0)	After Dewatering of Saturated Slope (h_u=1.0)	Dewatering Need (h_u=1.0)	Saturated Slope (h_u=0.6)	After Dewatering of Saturated Slope (h_u=0.6)	Dewatering Need (h_u=0.6)
Section 1	1.54	1.12	1.23	38 m	1.27	N/A	N/A
Section 2	1.99	1.45	N/A	N/A	1.64	N/A	N/A
Section 3	2.34	1.78	N/A	N/A	2.00	N/A	N/A
Section 4	1.71	1.26	N/A	N/A	1.42	N/A	N/A

Source: Koza, 2012

The results of the geotechnical analysis indicate that the interramp angle of 49° without ramps for section 1 is operationally stable in dry conditions but not so in saturated conditions. By drawing down the groundwater table by 38 m the factor of safety can be increased to 1.23 which is considered to be acceptable. Other sections are influenced by ramps reducing the overall angle from the inter-ramp angle and also other natural benefits due to topography and angle of the orebody.

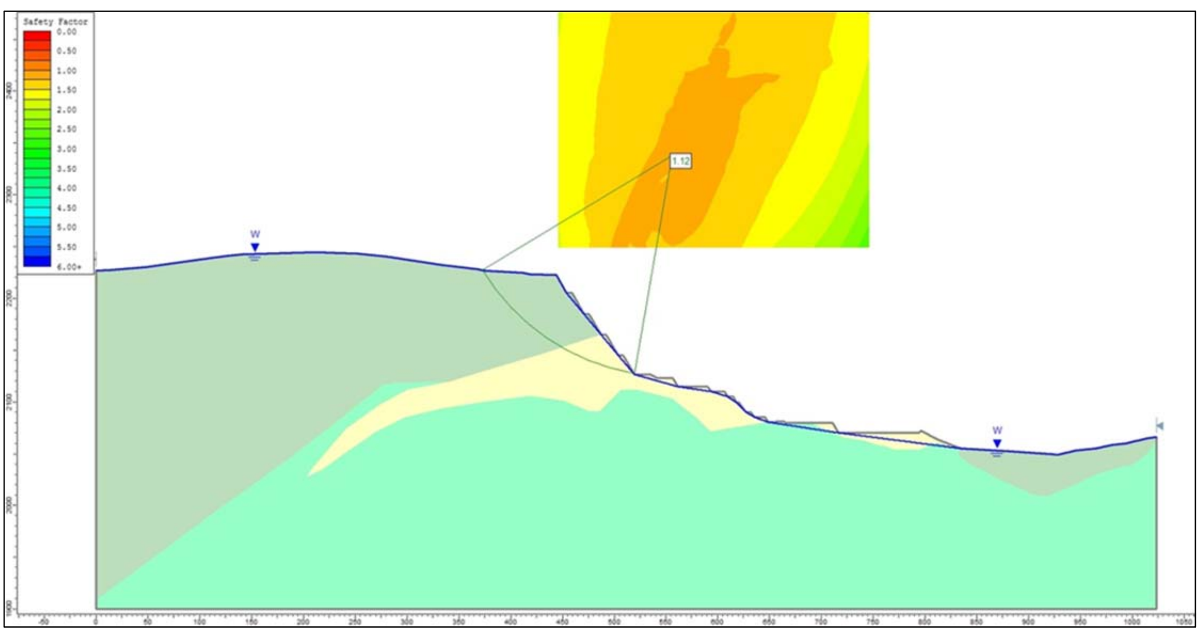
Figure 2.8.4 illustrates the high risk wall of section 1 that will require partial de-watering (40% head loss) to increase the factor of safety to acceptable but still risky limits.



Source: Koza, 2012

Figure 2.8.4: Section 1 Slope Stability with Partial Dewatering

If saturated groundwater conditions were modeled, the factor of safety will fall to an estimated factor of safety approaching 1.12 as illustrated in Figure 2.8.5. Although considered mineable, it is not recommended to go below a factor of safety of 1.3.



Source: Koza, 2012

Figure 2.8.5: Section 1 Slope Stability during Saturated Conditions

2.9 Metallurgy, Process Plant and Infrastructure

2.9.1 Introduction

Extensive metallurgical investigations have been conducted by McClelland Laboratories, Inc. (McClelland) on drill core samples from Mollakara. This work included initial bottle roll variability testing on 35 composites selected to represent calc-schist (CCS) and volcanic epiclastic (VVC) lithologies in both the oxide and transition zones of the deposit. The results of this variability testwork were used to formulate six test composites (2 from the transition zone and 4 from the oxide zone) for column leach testwork at P_{80} 32 mm and P_{80} 9.5 mm crush sizes. The details for these metallurgical investigations are presented in two separate reports issued by McClelland:

"Ore Variability Testing – Diyadin Oxide Heap Leach Drill Core Composites", McClelland Laboratories, November 24, 2010; and

"Heap Leach Cyanidation Testing – Mollakara Drill Core Composites", McClelland Laboratories, December 19, 2011.

2.9.2 Variability Metallurgical Investigations - McClelland 2010 Newmont

Bottle roll variability testwork was undertaken on 35 core samples of oxide and transition material crushed to 80% < 1.7 mm. Two ore types were identified:

- CSS – calc-silicate schist; and
- VVC – volcanics.

The testwork also identified different levels of clay alteration in the samples.

Results are summarized in Table 2.9.2.1.

Table 2.9.2.1: Summary Results – Scoping Bottle Roll Tests on Mollakara Drill Core Samples

Composites	Head Analysis			Au	g/t Au			Reagent Requirement	
	Au	S=	As	Recovery	Calculated			kg/Mt ore	
	g/Mt	%	%	%	Ext'd.	Tail	Head	NaCN Cons	Lime Added
Oxide CSS (16)									
Average	0.96	0.11	0.31	77.1	0.75	0.19	0.94	0.32	2.6
Maximum	2.28	0.41	2.33	88.1	1.72	2.15	2.15	2.49	4.1
Minimum	0.26	0.02	0.02	50.0	0.16	0.05	0.29	0.01	1.9
Oxide VVC (11)									
Average	0.87	0.11	0.4	85.2	0.77	0.11	0.88	0.21	3.2
Maximum	2.11	0.53	1.85	95.4	1.85	0.23	2.04	0.28	6.9
Minimum	0.22	0.01	0.01	75.3	0.17	0.04	0.22	0.08	1.5
Transitional CCS (6)									
Average	1.45	0.85	0.61	66.7	1.04	0.38	1.42	2.05	8.1
Maximum	2.51	2.93	1.3	90.2	2.29	0.69	2.54	6.17	18.1
Minimum	0.47	0.06	0.02	47.8	0.22	0.22	0.46	0.07	2
Transitional VVC (2)									
Average	0.84	0.12	0.21	77.4	0.6	0.25	0.85	0.15	2.1
Maximum	1.31	0.16	0.40	87.9	0.91	0.45	1.36	0.15	2.7
Minimum	0.36	0.07	0.02	66.9	0.29	0.04	0.33	0.15	1.4

It was concluded that the oxide zone composites generally responded very well to direct agitated cyanidation treatment. A single oxide zone composite displayed a below trend gold recovery (50.0%) and above trend cyanide consumption (2.49 kg/t ore). The 26 other oxide zone composites would be considered amenable to direct agitated cyanidation treatment, at the 1.7 mm feed size. Gold recoveries from those 26 composites ranged from 66.1 to 95.4%, and averaged 81.6%, after 96 hours of leaching. Corresponding cyanide consumptions and lime requirements averaged 0.16 kg/t ore and 2.9 kg/t ore, respectively.

The transitional zone composites were more varied in their response to cyanidation treatment. Three of the transitional zone composites (C1, C3 and C5) displayed below trend gold recoveries (47.8 to 56.2%), and high cyanide consumptions (2.77 to 6.17 kg/t ore) and high lime consumptions (9.5 to 18.1 kg/t ore). Gold recoveries from the remaining five transitional zone composites ranged from 66.9 to 90.2%, and averaged 79.9%, in 96 hours of leaching. Corresponding cyanide consumptions and lime requirements averaged 0.16 kg/t ore and 2.6 kg/t ore, respectively.

2.9.3 Column Leach Testwork – McClelland 2011

Detailed heap leach cyanidation testing was conducted on six drill core composites, which were prepared according to oxidation type (oxide or transitional), lithology and gold grade. Composites 1 and 2 represented transitional ore types, and composites 3 through 6 represented oxide ore types. Average head grades for the composites varied from 0.40 to 1.47 g/t Au. Sulfide sulfur content ranged from 0.18 to 0.85%. Sulfate sulfur content ranged from 0.36 to 1.08%. None of the composites contained greater than 0.1% organic carbon. The oxide ore type composites were observed to have a significant clay component. Column percolation leach tests were conducted on each composite at simulated secondary crusher discharge (P_{80} 32 mm) and tertiary crusher discharge (P_{80} 9.5 mm) feed sizes. Comparative bottle roll tests were conducted on each composite at an 80% -1.7 mm feed size.

<u>Composite Make-up and Head Analyses</u>

A total of 470 drill core interval samples were received for compositing and subsequent metallurgical testing. Each sample was combined, according to instructions provided by Koza personnel, to produce six drill core composites. The composites were identified as being either transitional ore type (Composites C-1 and C-2) or oxide ore type (Composites C-3 through C-6). Air dried composites were stage-crushed to P_{80} -32 mm (100% -50 mm) and thoroughly blended by repeated coning and were quartered to obtain approximately 125 kg for a column leach test and 25 kg for a head screen analysis. The remaining -32 mm material from each composite was stage-crushed to minus 19 mm in size. The minus 19 mm composite material was thoroughly blended and split to obtain 10 kg for generation of an abrasion index test sample, and 110 kg for finer crushing. Each 110 kg split was stage crushed to P80 -9.5 mm (100% -12.5 mm), thoroughly blended, and split to obtain approximately 70 kg for a column leach test, 15 kg for a head screen analysis, four 1 kg samples for head assay, 10 kg for agglomeration testing and 10 kg for finer crushing. Each 10 kg split for finer crushing was stage crushed to 80% -1.7 mm for bottle roll testing and mineralogy.

Head samples were assayed using conventional fire assay fusion procedures to determine gold content. A four acid-digestion/Atomic Absorption (AA) finish procedure was used to determine silver content. A single head sample from each composite was also submitted for a multi-element ICP analysis, a “classical whole rock” analysis, sulfur speciation (total, sulfide and sulfate) analyses and carbon speciation (total, organic and inorganic) analyses. Head assay results and head grade comparisons are presented in Table 2.9.3.1. The results from ICP scan analyses are presented in Table 2.9.3.2 and results from “classical whole rock”, carbon and sulfur speciation analyses are presented in Table 2.9.3.3.

Table 2.9.3.1: Head Assays for Mollakara Drill Core Composites

Head Grade, g Au/mt ore						
Determination	**C-1**	**C-2**	**C-3**	**C-4**	**C-5**	**C-6**
Direct Assay, Init.	0.38	1.04	0.90	0.38	0.94	1.45
Direct Assay, Dup.	0.36	1.08	0.83	0.46	1.14	1.40
Direct Assay, Trip.	0.39	1.14	0.82	0.50	0.92	1.34
Calc'd., Bottle Roll, 1.7 mm	0.43	1.17	1.09	0.55	1.11	1.57
Calc'd., Head Screen, 31.5 mm	0.43	1.27	0.89	0.49	1.02	1.48
Calc'd., Head Screen, 9.5 mm	0.39	1.15	0.86	0.52	0.99	1.50
Calc'd., Column, 31.5 mm	0.42	1.19	0.87	0.51	0.98	1.50
Calc'd., Column, 9.5 mm	0.38	1.07	0.86	0.48	0.99	1.51
Average	**0.40**	**1.14**	**0.89**	**0.49**	**1.01**	**1.47**
Std. Deviation	0.03	0.07	0.09	0.05	0.08	0.07
Precision, %	92.5	93.9	89.9	89.8	92.1	95.2

Source: McClelland, 2011

Table 2.9.3.2: ICP Metals Analyses - Mollakara Drill Core Composites

Sample Analysis	Unit	C-1	C-2	C-3	C-4	C-5	C-6
Ag	mg/kg	0.17	0.10	0.28	0.16	0.19	0.21
Al	%	3.12	3.37	6.70	4.45	4.90	4.83
As	mg/kg	3,680	6,720	4,820	5,170	3,850	6,180
Ba	mg/kg	670	560	1,070	790	950	1,030
Be	mg/kg	0.53	0.59	0.97	0.68	0.75	0.83
Bi	mg/kg	0.12	0.15	0.26	0.19	0.24	0.22
Ca	%	10.60	6.60	1.65	8.77	5.85	4.05
Cd	mg/kg	0.29	0.33	0.30	0.19	0.12	0.48
Ce	mg/kg	35.7	44.4	82.9	47.0	64.9	66.1
Co	mg/kg	6.6	6.6	4.8	7.6	5.1	6.5
Cr	mg/kg	29	29	64	34	39	50
Cs	mg/kg	8.89	8.68	13.80	8.61	8.97	12.75
Cu	mg/kg	21.1	21.6	29.7	22.8	28.9	21.6
Fe	%	1.63	1.56	1.72	1.71	1.33	1.90
Ga	mg/kg	7.12	8.10	18.50	7.46	9.67	10.90
Ge	mg/kg	0.11	0.13	0.13	0.09	0.11	0.12
Hf	mg/kg	0.3	0.3	1.0	0.5	0.7	0.8
Hg	mg/kg	9.9	17.6	21.4	13.7	22.3	20.3
In	mg/kg	0.038	0.052	0.080	0.042	0.062	0.069
K	%	0.44	0.39	0.74	0.34	0.39	0.54
La	mg/kg	17.4	21.3	40.0	21.5	30.0	29.7
Li	mg/kg	19.0	22.9	38.1	29.0	33.4	29.1
Mg	%	5.43	3.03	0.56	3.62	2.32	1.57
Mn	mg/kg	923	907	361	896	616	627
Mo	mg/kg	4.25	2.11	2.46	1.72	1.77	1.97
Na	%	0.03	0.04	0.16	0.05	0.05	0.06
Nb	mg/kg	5.6	6.1	8.9	5.7	7.2	8.0
Ni	mg/kg	21.6	23.4	29.5	27.6	27.2	27.2
P	mg/kg	390	500	760	700	730	820
Pb	mg/kg	16.6	14.6	25.2	13.6	19.2	20.0
Rb	mg/kg	27.4	24.9	43.9	27.7	30.4	41.2
Re	mg/kg	0.007	0.006	<0.002	0.003	0.003	0.003
S	%	2.00	1.82	1.03	1.33	1.07	1.16
Sb	mg/kg	45.9	76.1	71.4	57.8	71.0	127.0
Sc	mg/kg	3.8	5.3	8.5	5.8	8.3	6.7
Se	mg/kg	3	5	3	2	3	4
Sn	mg/kg	1.1	1.3	2.2	1.4	3.8	2.0
Sr	mg/kg	146.5	200	368	253	321	301
Ta	mg/kg	0.40	0.46	0.70	0.50	0.57	0.62
Te	mg/kg	0.90	2.13	2.42	0.95	1.94	3.36
Th	mg/kg	6.0	7.8	14.1	7.2	9.1	9.6
Ti	%	0.145	0.174	0.245	0.161	0.195	0.224
Tl	mg/kg	107.0	99.0	86.3	55.8	63.9	113.5
U	mg/kg	3.2	2.5	4.6	2.2	3.3	3.6
V	mg/kg	52	49	120	58	69	81
W	mg/kg	5.1	8.2	12.0	8.5	9.8	13.3
Y	mg/kg	10.4	10.8	15.2	17.5	16.2	16.6
Zn	mg/kg	54	50	35	51	36	69
Zr	mg/kg	8.9	10.4	31.9	13.2	22.9	25.7

Source: McClelland, 2011

Table 2.9.3.3: Whole Rock, Carbon and Sulfur Speciation Analyses, Mollakara Drill Core Composites

Sample Analyte	Unit	C-1	C-2	C-3	C-4	C-5	C-6
Al_2O_3	%	5.37	5.98	12.85	7.95	9.17	8.86
BaO	%	0.13	0.09	0.13	0.10	0.12	0.13
CaO	%	15.45	9.58	2.31	12.50	8.63	5.74
Cr_2O_3	%	<0.01	<0.01	0.01	0.01	0.01	0.01
Fe_2O_3	%	2.24	2.18	2.44	2.46	1.97	2.81
K_2O	%	0.50	0.47	0.92	0.42	0.50	0.69
MgO	%	8.68	4.95	0.96	5.95	3.99	2.64
MnO	%	0.12	0.12	0.05	0.12	0.08	0.08
Na_2O	%	0.08	0.10	0.25	0.11	0.11	0.14
P_2O_5	%	0.09	0.13	0.21	0.17	0.18	0.24
SiO_2	%	41.8	57.7	68.5	46.2	58.8	64.5
SrO	%	0.02	0.02	0.05	0.03	0.04	0.03
TiO_2	%	0.29	0.35	0.49	0.33	0.42	0.44
LOI	%	18.75	13.20	10.55	16.15	11.55	8.09
Total	**%**	**93.5**	**94.9**	**99.7**	**92.5**	**95.6**	**94.4**
C (Total)	%	5.73	3.43	0.80	4.23	2.87	1.93
C (Organic)	%	0.03	0.04	0.06	0.07	0.05	0.11
C (Inorganic)	%	5.34	3.17	0.63	3.87	2.59	1.68
S (Total)	%	1.73	1.66	0.99	1.26	1.04	1.12
S (Sulfate)	%	0.67	0.77	0.36	1.08	0.68	0.70
S (Sulfide)	%	0.85	0.68	0.68	0.18	0.25	0.45
CO_2	%	19.6	11.6	2.3	14.2	9.5	6.2

Source: McClelland, 2011

Of particular note, mercury content in the test composites ranged from 9.9 to 22.3 mg/kg Hg. At these mercury levels it can be anticipated that a retort will be required to process the precious metal precipitates prior to smelting. Carbon speciation results showed that none of the composites contained greater than 0.1% organic carbon, indicating that preg-robbing should not be a problem. Sulfur speciation results showed that the composites contained between 0.18% and 0.85% sulfide sulfur. Both the transitional ore type and oxide ore type composites contained significant quantities of sulfate sulfur (0.36 to 1.08% S). X-ray diffraction (XRD) analysis results showed that all of the composites contained elevated concentrations of kaolinite ($Al_2Si_2O_5(OH)_4$), a clay mineral. The two transitional ore type composites contained 14 to 17% kaolinite. The four oxide composites contained between 19% and 28% kaolinite. The high clay content in these composites indicates that agglomeration prior to heap leaching will be important.

Bottle Roll Cyanidation Testwork

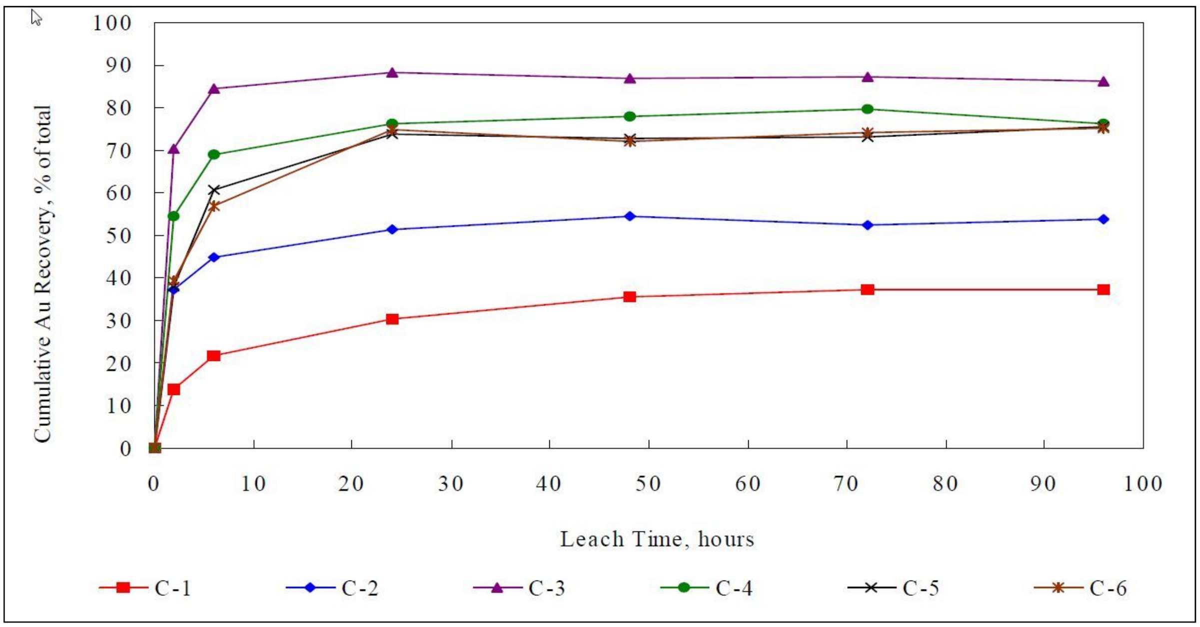
Bottle roll cyanidation tests were conducted on each of the Mollakara composites at an 80% -1.7 mm feed size to determine ultimate gold recovery and reagent requirements. Bottle roll tests were conducted at a slurry density of 40% solids with a cyanide concentration of 1 g/t NaCN and the slurry pH maintained at about 11.0 with lime. Tests were conducted for a total of 96 hours with solutions samples taken after 2, 6, 24, 48, and 72 hours.

Overall bottle roll results are provided in Table 2.9.3.4 and gold leach profiles are presented graphically in Figure 2.9.3.1. Gold extraction rates were fairly rapid and substantially complete after 24 hours of leaching. Cyanide consumptions for the transitional ore type composites were fairly high at about 1.50 kg/t ore. The cyanide consumption and lime demand data may indicate the presence of reactive sulfide minerals in the transitional ore type composites. Cyanide consumptions for the oxide ore type composites were substantially lower at 0.22 to 0.75 kg/t ore. Lime requirements for the oxide ore type composites ranged from 3.4 to 5.2 kg/t ore.

Table 2.9.3.4: Bottle Roll Test Results, Mollakara Drill Core Composites

Cumulative Au Extraction %	C-1	C-2	C-3	C-4	C-5	C-6
in 2 hours	14.0	37.2	70.2	54.5	37.8	39.2
in 6 hours	21.9	44.8	84.5	69.1	60.6	57.1
in 24 hours	30.2	51.4	88.4	76.1	73.9	75.0
in 48 hours	35.6	54.5	86.8	77.9	72.9	71.9
in 72 hours	37.2	52.4	87.4	79.5	73.0	74.0
in 96 hours	**37.2**	**53.8**	**86.2**	**76.4**	**75.7**	**75.2**
Calc'd. Head, g Au/mt ore	0.43	1.17	1.09	0.55	1.11	1.57
Assayed Head, g Au/mt ore(1)	0.37	1.09	0.85	0.45	1.00	1.40
NaCN Consumed, kg/mt ore	1.50	1.49	0.22	0.28	0.59	0.75
Lime Added, kg/mt ore	8.3	9.4	3.4	4.1	5.2	4.6
Final pH	11.2	11.1	11.0	11.1	11.2	11.0
Natural pH (40% solids)	6.3	6.2	7.7	7.5	7.5	6.9

(1) Average of all triplicate assays.



Source: McClelland, 2011

Figure 2.9.3.1: Bottle Roll Gold Extraction versus Retention Time

Agglomeration Strength and Stability Testing

Agglomerate strength and stability tests were conducted on three composites (C-1, C-3 and C-5) at the P_{80} -9.5 mm crush size to optimize agglomerating conditions. A 10 kilogram split of feed was dry screened at 1.7 mm to determine the natural quantity of plus 1.7 mm material. Both plus and minus 1.7 mm screened material were then recombined on a weighted basis to produce ten identical one kilogram splits for agglomerate strength and stability tests. Ore charges were agglomerated by adding the appropriate quantity of binder, wetting with water to the optimum moisture content (determined visually) and curing in sealed containers for 72 hours before being subjected to the "jigging" test.

Prepared agglomerates were placed onto a 1.7 mm screen and were "jigged" in and out of a container of water 10 times in a 30 second period. Jigging in this manner imparts a shear stress to the agglomerates substantially more severe than that imparted by percolating solution. Stability was measured empirically by comparing the quantity of agglomerates retained on a 1.7 mm screen after jigging with the quantity of feed naturally retained on a 10 mesh screen (dry screening). Stable agglomerates are usually produced when over a 30 weight percent improvement in the quantity of feed retained on the screen after "jigging" is achieved.

Agglomerate strength tests were conducted by selecting two typical agglomerates from each agglomerated charge before jigging and submerging them in separate beakers of water and observing the degree of agglomerate degradation in a 24 hour period. An agglomerate with sufficient grain strength to overcome swelling tendencies of contained clays would not degrade in 24 or more hours of complete submersion. Complete degradation means that the submerged agglomerate broke down to a natural state within 10 minutes of submersion. Optimum agglomerating conditions were determined by the point at which near maximum weight percentage was retained on the 1.7 mm screen and the point at which no degradation occurred within 24 hours of submersion. Agglomerate strength and stability test results are presented in Table 2.9.3.5. Optimum agglomeration binder additions were determined to be lime, equivalent to 80% of the bottle roll test lime requirement, plus cement, equivalent to 2.0 kg/t ore for the transitional ore type, and 3.0 kg/t ore for the oxide ore type. Optimum agglomeration moisture (determined visually) was approximately 8% for the transitional ore type material, and 12% for the oxide ore type material. These optimum agglomerating conditions were used for the 9.5 mm feed size column test feeds. Cement addition for agglomeration of the minus 32 mm crush size material was decreased to 1.0 kg/mt ore.

Table 2.9.3.5: Agglomerate Strength and Stability Tests on Mollakara Composites

Composite	Binder Addition, kg/mt ore		Moisture%	Retained on 1.7mm Screen, Wt. %	Submersion Observation (degree of degradation)	
	Cement	Lime			10 Minutes	24 Hours
Comp C-1	N/A	N/A	0.0	62.6	Dry Screened	
Comp C-1	0.0	0.0	7.6	82.9	Partial	Partial
Comp C-1	0.0	8.3	8.3	88.8	Partial/None	Partial/None
Comp C-1	1.0	6.6	8.7	88.6	Partial/None	Partial/None
Comp C-1	2.0	6.6	8.5	88.6	None	None
Comp C-1	6.0	0.0	8.0	88.3	Partial/None	Partial
Comp C-1	8.0	0.0	8.1	90.5	None	None
Comp C-1	10.0	0.0	7.9	91.8	None	None
Comp C-1	12.5	0.0	7.8	94.6	None	None
Comp C-3	N/A	N/A	11.7	54.2	Dry Screened	
Comp C-3	0.0	3.4	12.7	90.8	Partial/None	Total
Comp C-3	1.0	2.7	13.0	91.2	Partial/None	Total
Comp C-3	2.0	2.7	12.1	92.2	Partial/None	Partial
Comp C-3	3.0	0.0	12.1	91.0	Partial/None	Total
Comp C-3	4.0	0.0	12.3	90.8	Partial/None	Total
Comp C-3	5.0	0.0	11.4	92.4	Partial	Total
Comp C-3	7.5	0.0	11.2	93.0	Partial/None	Total
Comp C-3	3.0	2.7	12.4	90.3	None	Partial/None
Comp C-3	5.0	2.7	12.7	94.5	None	Partial/None
Comp C-5	N/A	N/A	9.4	61.3	Dry Screened	
Comp C-5	0.0	5.2	11.8	92.4	Partial/None	Total
Comp C-5	1.0	4.2	11.4	95.1	Partial/None	Total
Comp C-5	2.0	4.2	11.0	91.1	Total	Total
Comp C-5	3.0	0.0	9.6	91.0	Partial/None	Total
Comp C-5	4.0	0.0	9.7	93.6	Partial/None	Total
Comp C-5	5.0	0.0	10.4	95.3	Total	Total
Comp C-5	7.5	0.0	10.6	95.5	Partial	Total
Comp C-5	3.0	4.2	11.7	91.0	None	None
Comp C-5	5.0	4.2	12.1	92.2	None	None

Source: McClelland, 2011

Column Leach Testwork

Column percolation leach tests were conducted on each of the six composites at P_{80} -32 mm and P_{80} -9.5 mm crush sizes to determine gold extraction, extraction rate, reagent requirements and feed size sensitivity, under simulated heap leaching conditions. The ore charges were agglomerated by adding the appropriate quantity of lime and cement, wetting with water to optimum moisture content (determined visually), mechanically tumbling to affect agglomeration, and curing in 3 m high leaching columns before applying leach solution. Agglomerates were placed into the columns in a manner to minimize particle segregation and compaction. Column diameters used for the 32 mm and 9.5 mm feeds were 20 cm and 15 cm, respectively.

Leaching was conducted by applying cyanide solution at a concentration of 1.0 g/L NaCN at a rate of 0.20 Lpm/m^2 (0.005 gpm/ft^2). Pregnant effluent solutions were collected for each 24 hour period. Pregnant solution volumes were measured by weighing, and samples were taken for gold and silver analysis using conventional AA methods. Cyanide concentration and pH were determined for each pregnant solution. Pregnant solutions were pumped through a three stage carbon circuit for adsorption of dissolved gold values. Barren solution, with appropriate make-up reagent, was applied to the ore charges daily. After leaching, water washing was conducted to remove residual cyanide and to recover dissolved gold values. Moisture required to saturate the ore charges (in process solution inventory), for agglomeration and retained moistures were determined. Apparent ore bulk densities were measured before and after leaching. Drain down tests were conducted after rinsing was complete.

After leaching, rinsing, and draining, residues were removed from the columns and moisture samples were taken immediately. A “split” of moist agglomerates was also taken from each column residue for load/permeability testing. Remaining leached residues were air dried, blended and split to obtain a sample for a tail screen analysis.

Overall metallurgical results from column tests are shown in Table 2.9.3.6 and Table 2.9.3.7 and gold leach rate profiles are shown graphically in Figures 2.9.3.2 and 2.9.3.3. Physical ore characteristics data are provided in Table 2.9.3.8.

Table 2.9.3.6: Column Test Results, Mollakara Transitional Ore Type

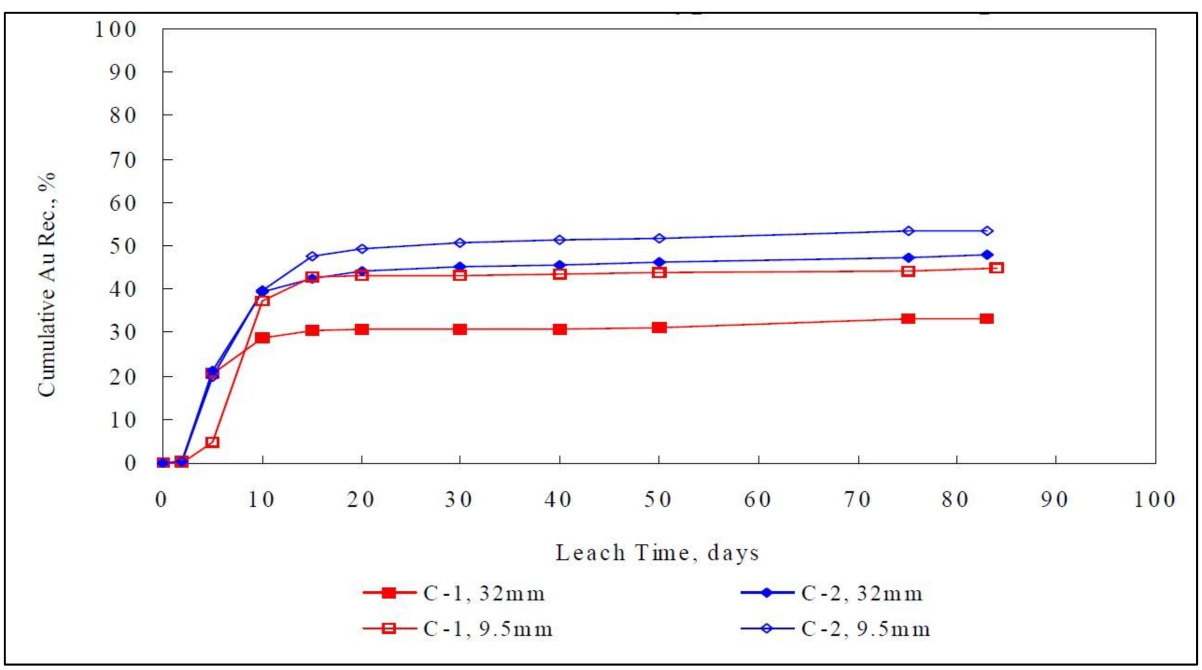
Metallurgical Results	**C-1**		**C-2**	
Extraction: % of Total Au	**32 mm**	**9.5 mm**	**32 mm**	**9.5 mm**
1st Effluent	0.2	0.0	0.3	0.3
in 5 days	20.7	4.7	21.3	19.7
in 10 days	28.9	37.2	39.3	39.8
in 15 days	30.4	42.8	42.6	47.6
in 20 days	30.9	43.3	44.1	49.4
in 30 days	30.9	43.3	45.3	50.7
in 40 days	30.9	43.6	45.7	51.3
in 50 days	31.6	43.9	46.4	51.6
in 75 days	33.3	44.1	47.4	53.3
End of Leach/Rinse Cycle	33.3	44.7	47.9	53.3
Extracted, g Au/mt ore	0.14	0.17	0.57	0.57
Tail Screen, g Au/mt ore(1)	0.28	0.21	0.62	0.50
Calc'd Head, g Au/mt ore	0.42	0.38	1.19	1.07
Average Head, g Au/mt ore(2)	0.40	0.40	1.14	1.14
NaCN Consumed, kg/mt ore	1.38	1.85	1.83	2.04
Lime Added, kg/mt ore	6.6	6.6	7.5	7.5
Cement Added, kg/mt ore	1.0	2.0	1.0	2.0
Ag Recovery, % of total	N/A	N/A	N/A	N/A
Extracted, g Ag/mt ore	<0.1	<0.1	<0.1	<0.1
Tail Screen, g Ag/mt ore(1)	0.4	0.4	1.0	0.4
Calc'd Head, g Ag/mt ore	<0.5	<0.5	<1.1	<0.5
Average Head, g Ag/mt ore(2)	0.3	0.3	0.6	0.6
Final Solution pH	10.8	10.7	10.2	10.2
pH After Rinse	9.8	8.9	9.3	9.3
Leach/Rinse Cycle, Days	83	84	83	83

Source: McClelland 2011
(1) Average of all head grade determinations
(2) Average of all triplicate direct assays

Table 2.9.3.7: Column Test Results - Mollakara Oxide Ore Type

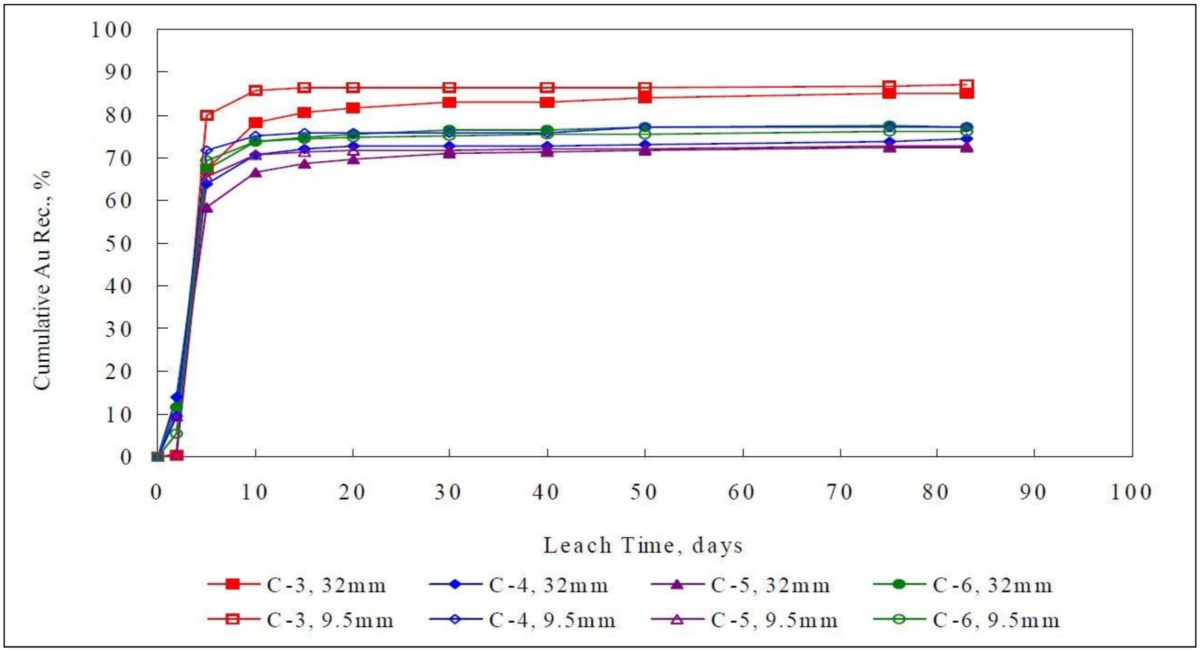
Metallurgical Results	C-3		C-4		C-5		C-6	
Extraction: % of total Au	32 mm	9.5 mm	32 mm	9.5 mm	32 mm	9.5 mm	32 mm	9.5 mm
1st Effluent	0.3	0.2	14.0	9.5	9.6	0.7	11.6	5.4
in 5 days	67.2	80.0	63.9	71.7	58.5	65.4	67.2	69.2
in 10 days	78.2	85.7	70.5	75.1	66.7	70.8	73.6	73.8
in 15 days	80.4	86.2	72.0	75.7	68.7	71.4	74.9	74.4
in 20 days	81.4	86.3	72.6	75.9	69.7	71.6	75.5	74.7
in 30 days	82.8	86.3	72.6	75.9	70.9	71.8	76.4	75.0
in 40 days	83.1	86.3	72.6	75.9	71.3	71.9	76.6	75.3
in 50 days	83.8	86.4	73.1	77.1	71.7	72.1	77.0	75.5
in 75 days	84.9	86.7	73.6	77.1	72.4	72.7	77.3	76.2
End of Leach/Rinse Cycle	85.1	87.2	74.5	77.1	72.4	72.7	77.3	76.2
Extracted, g Au/mt ore	0.74	0.75	0.38	0.37	0.71	0.72	1.16	1.15
Tail Screen, g Au/mt ore(1)	0.13	0.11	0.13	0.11	0.27	0.27	0.34	0.36
Calc'd Head, g Au/mt ore	0.87	0.86	0.51	0.48	0.98	0.99	1.50	1.51
Average Head, g Au/mt ore(2)	0.89	0.89	0.49	0.49	1.01	1.01	1.47	1.47
NaCN Consumed, kg/mt ore	1.02	1.36	1.10	1.29	1.12	1.39	1.24	1.50
Lime Added, kg/mt ore	2.7	2.7	3.3	3.3	4.2	4.2	3.7	3.7
Cement Added, kg/mt ore	1.0	3.0	1.0	3.0	1.0	3.0	1.0	3.0
Ag Recovery, % of total	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Extracted, g Ag/mt ore	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Tail Screen, g Ag/mt ore(1)	0.3	0.4	0.3	0.4	0.4	0.4	0.4	0.4
Calc'd Head, g Ag/mt ore	<0.4	<0.5	<0.4	<0.5	<0.5	<0.5	<0.5	<0.5
Average Head, g Ag/mt ore(1)	0.4	0.4	0.3	0.3	0.4	0.4	0.3	0.3
Final Solution pH	11.0	11.1	10.6	10.8	10.7	10.7	10.5	10.6
pH After Rinse	10.7	11.0	9.8	9.7	9.8	9.7	9.6	9.7
Leach/Rinse Cycle, Days	83	83	83	83	83	83	83	83

Source: McClelland 2011
(1) Average of all head grade determinations.
(2) Average of all triplicate direct assays.



Source: McClelland, 2011

Figure 2.9.3.2: Leach Curve for Mollakara Transition Ore Composite



Source: McClelland, 2011

Figure 2.9.3.3: Leach Curve for Mollakara Oxide Ore Composite

Table 2.9.3.8: Physical Data from Mollakara Column Tests

Sample Designation	Feed Size (mm)	Test No.	Ore Charge kg	Moisture, wt. %				Apparent Bulk Density Mt ore/m^3	
				As Rec'd.	For Agglomeration	To Saturate*	Retained	Before	After
C-1	32	P-1	116.27	0.1	7.5	16.1	8.3	1.35	1.36
C-1	9.5	P-7	65.38	1.8	10.5	25.5	9.6	1.31	1.31
C-2	32	P-2	116.02	0.1	8.7	18.8	10.2	1.27	1.28
C-2	9.5	P-8	65.22	0.2	13.1	28.8	12.5	1.19	1.19
C-3	32	P-3	116.33	3.1	14.6	29.4	16.2	1.10	1.21
C-3	9.5	P-9	65.45	0.5	14.2	33.9	14.9	1.14	1.17
C-4	32	P-4	119.37	0.6	8.7	17.3	9.0	1.27	1.28
C-4	9.5	P-10	65.47	0.1	10.2	23.9	10.4	1.28	1.28
C-5	32	P-5	119.87	1.1	9.7	20.7	9.4	1.15	1.15
C-5	9.5	P-11	64.69	0.3	10.7	26.0	12.5	1.16	1.16
C-6	32	P-6	111.30	0.1	9.6	22.2	11.2	1.26	1.27
C-6	9.5	P-12	60.60	0.9	11.7	23.6	9.0	1.14	1.14

Source: McClelland 2011
* Calculated on a dry ore weight basis, includes moisture for agglomeration.

Results from these column tests indicated that the transitional ore type composites were not readily amenable to simulated heap leach cyanidation treatment at the 32 mm and 9.5 mm feed sizes. Gold recoveries obtained from composites C-1 and C-2, at the P_{80} -32 mm crush size, were 33.3% and 47.9%, respectively after 83 days of leaching and rinsing. Crushing the composites to P_{80} -9.5 mm improved respective gold recoveries to 44.7% and 53.3% after 83 days. Gold extraction was substantially complete after 20 days of leaching. Additional gold values were extracted at a much slower rate after 20 days. Cyanide consumptions were fairly high at 1.38 and 1.83 kg/t ore for the transition composites crushed to P_{80} 32 mm. Crushing to P_{80} -9.5mm increased respective cyanide consumptions to 1.85 - 2.04 kg/t. Lime at 6.6 to 7.5 kg/t ore and cement at 1.0 to 2.0 kg/t added during agglomeration pretreatment were sufficient to maintain protective alkalinity during leaching.

Column tests on the oxide composites demonstrated that this ore type was readily amenable to simulated heap leach cyanidation treatment, at both crush sizes (P_{80} -32 mm and P_{80} -9.5 mm) and gold extraction was not sensitive to the feed sizes tested. The highest gold extractions were achieved from composite C-2, which was the only oxide ore type composite comprised of core with the "VVC" lithology code. Gold extractions obtained from composite C-2 at the 80% -32 mm and 80% -9.5 mm feed sizes were 85.1 and 87.2%, respectively. Gold extractions achieved from the CCS lithology code composites (C-4 and C-5) and the CSS/VVC blend (C-6), at the 80% -32 mm feed size, ranged from 72.4 to 77.3%. Gold extractions from the corresponding 80% -9.5 mm crush sizes ranged from 72.7 to 77.1%, and were considered to be essentially the same as achieved at the 32 mm crush size. Gold extraction rates were very rapid, and gold extraction was substantially complete after 10 days of leaching. Additional gold values were extracted after an additional 10 days, but at a very slow rate.

Cyanide consumptions with the oxide ore composites were moderate, and ranged from 1.02 to 1.24 kg/t (average 1.12 kg/t) at the P_{80} -32 mm crush size. Cyanide consumptions at the P_{80} -9.5 mm crush size averaged 1.39 kg/t. It should be noted that column test cyanide consumptions are usually substantially higher than experienced during commercial heap leaching, for relatively "clean" oxide ores. Commercial consumption for the Mollakara oxide ore type material should be substantially lower, and probably would not exceed 0.8 kg/t ore. Lime at 2.7 to 4.2 kg/t ore and cement at 1.0 to 3.0 kg/t ore added during agglomeration pretreatment, were sufficient to maintain protective alkalinity during leaching.

Physical ore characteristic data show that very little slumping of the agglomerated ore occurred during leaching. Bulk densities were essentially the same before and after leaching. This is unusual for agglomerated ore. It is expected that significant "slumping" of the agglomerates will occur during commercial heap leaching of the material represented by the composites tested. Moisture requirements were high, particularly for composite C-3. Moisture requirements tended to increase with increasing ore fines content. However, no solution percolation, fines migration or solution channeling problems were encountered during leaching.

Conclusions

- The Mollakara oxide ore type material was readily amenable to simulated heap leach cyanidation treatment, at an 80% -32 mm feed size;
- Crushing finer (80% -9.5 mm) in size did not result in a significant improvement in gold recovery from the oxide ore type material;
- Gold recovery rates were very rapid, which may make heap leaching on a re-useable ("on/off") type heap a viable option;
- Cyanide consumptions for the oxide ore type material were low to moderate, and are not expected to exceed 0.8 kg/t ore, in commercial production;
- The oxide ore type material generally contained a relatively high percentage of clay fines, and will require agglomeration pretreatment during commercial heap leaching. Binder additions equivalent to 80% of the lime required for pH control, along with 1.0 kg/t cement was found to be optimum agglomeration pretreatment of the oxide ore type;

- The high clay fines content of the oxide ore may present material handling difficulties during commercial crushing, agglomerating and heap leaching. In particular, difficulties can be expected if the high clay ore has a significant moisture content when fed to the crushing plant. After crushing, moisture content of the ore feeding the agglomerating circuit will need to be significantly lower than the indicated optimum agglomeration moisture, for successful agglomeration;
- The transitional ore type material was not as amenable to simulated heap leach cyanidation, and gold extraction was more sensitive to feed size;
- Crushing the transitional ore type material from P_{80} 32 mm to P_{80} 9.5 mm increased gold extraction by about 8%;
- Gold extraction for the transitional ore typed was rapid, but still slower than extraction rates observed with the oxide ore composites;
- Cyanide consumptions for the transitional ore types were higher than for the oxide ore types.
- The transitional ore typed may contain reactive sulfide minerals, which could have contributed to the higher cyanide consumptions; and
- The fines content of the transitional ore type was lower than for the oxide ore type, but was sufficiently high to require agglomeration pretreatment for commercial heap leaching.

2.9.4 Koza Column Testing

Metallurgical studies conducted by McClelland demonstrated equivalent gold extraction at the P_{80} 9.5 mm and P_{80} 32 mm crush sizes tested. These results indicated that leaching at a coarser crush size might be possible. To demonstrate this possibility, Koza ran additional column tests on bulk ore composites at both 25 mm and 90 mm crush sizes. The results of this work are reported by Koza in their report, “Koza Metallurgy Laboratory Report, Diyadin Ore Column Leach Tests”, October 23, 2012. The bulk test composite used for this program was developed from 21 surface samples representing both oxidized and transition zones in the Mollakara ore deposit. Samples were selected from different locations and excavated with a backhoe. Table 2.9.4.1 provides a description and locations of the samples used to form a 15 ton master bulk composite which was transported to Koza’s Kaymaz Metallurgical Laboratory.

Table 2.9.4.1: Sample Locations for the Bulk Mollakara Heap Leach Test Composite

SAMPLE LOCATION NUMBER	THE AMOUNT OF SCOOP	ZONE	LITH	X-COORDINATE	Y-COORDINATE	Z-COORDINATE	COMMENT
1	2	OX	VVC	377022.574	4362589.8	2121.06744	CLOSE TO DDD023
2	2	OX	VVC	376993.482	4362632.94	2130.05569	CLOSE TO DDD084
3	1	OX	VVC	376920.265	4362547.27	2149.11493	CLOSE TO DDD027
4	1	OX	VVC	376934.713	4362531.1	2146.68108	CLOSE TO DDD071,DDD059, DDD061
5	1	OX	CCS	376934.867	4362520.5	2145.06822	POINT OF DDD059
6	2	OX	CCS	376973.963	4362582.88	2141.61707	POINT OF DDD067
7	2	OX	CCS	376994.389	4362553.8	2130.32288	POINT OFDDD046
8	1	OX	CCS	377096.22	4362428.76	2131.6498	POINT OFDDD024,DDD032, DDD031
9	1	OX	CCS	377106.597	4362402.94	2124.57945	POINT OF DDD064
10	2	TR/QX	CCS	377247.077	4362371.05	2067.9182	POINT OF DDD075
11	1	OX	CDM	377209.432	4362278.73	2122.18021	POINT OF DDD052
12	1	OX	CDM	376442.057	4362694.02	2111.14789	CLOSE TO DDD052
13	1	TR	CCS	376739.811	4362938.88	2066.34864	CLOSE TO DDD102
14	1	OX	CCS	377097.167	4362451.37	2132.85284	POINT OF DDD024
15	1	OX	CCS	377094.745	4362431.36	2131.42454	POINT OF DDD032
16	1	OX	CCS	377093.503	4362365.62	2125.727	POINT OF DDD031
17	1	OX	VVC	376931.18	4362552.67	2152.80493	POINT OF DDD071
18	1	OX	VVC	377193.863	4362504.85	2077.27979	POINT OF DDD078
19	1	OX	VVC	376936.659	4362482.71	2144.67896	POINT OF DDD026
20	1	OX	VVC	377177	4362303	2122	POINT OF DDD009
21	1	OX	VVC	377193.201	4362446.2	2076.20093	POINT OF DDD074

Column Testing

Column testing of the bulk ore composite was conducted at both 90 mm and 25 mm crush sizes. To accomplish this, approximately 10 tons of Diyadin oxide ore was crushed to -90 mm using the primary jaw crusher at the Kaymaz process plant. The crushed sample was then blended and a portion was split out and crushed to -25 mm. In addition, a head sample was split out for chemical analysis. The resulting head analyses for the bulk composite are presented in Table 2.9.4.2. The gold head grade was reported at 0.9 g/t and silver head grade was reported at 1.04 g/t.

Table 2.9.4.2: Head Analyses - Mollakara Bulk Composite

Element	Unit	Value
Au	ppm	0.9
Ag	ppm	1.04
S	%	0.36
As	ppm	2,161
Sb	ppm	40
Cu	ppm	19.5
Ni	ppm	626
Fe	%	2.79
C	%	0.83
Co	ppm	69
Pb	ppm	19
Zn	ppm	88

Source: Koza, 2012

The 90 mm ore sample was leached in a 1m diameter x 6 m high steel column and the 25 mm ore sample was leached in a 0.5 m diameter x 6 m high column. The test conditions used for the column testwork are presented in Table 2.9.4.3. The initial cyanide leach solution concentration was 500 ppm and the application rate was 10 L/hr/m2 with the pH maintained at about 10.5. Leaching with cyanide was conducted over a 28 day cycle, followed by a 28 day wash cycle during which cyanide was not added. The results of the column test program are summarized in Table 2.9.4.4. At the 90 mm crush size 61.7% of the gold was extracted after 28 days of cyanide leaching. After the 28 day wash cycle a total of 67.5% of gold was extracted. This compares to 72.9% gold extraction at the -25 mm crush at the end of 28 days of cyanidation and 28 days of washing. Cyanide consumption was reported at 0.82 kg/t ore at the 25 mm crush size and 0.70 kg/t at the 90 mm crush size. Lime consumption was reported at 3.5 kg/t ore/.

Table 2.9.4.3: Summary of Column Test Conditions

Parameter	Units	Crush Size	
		25 mm	90 mm
Feed	kg	1,530	5,460
PH		10.5	10.5
Cyanide Concentration	ppm	500	500
Application Rate	L/hr/m2	10	10
Leach Duration	Days	28	28
Wash	Days	28	28

Table 2.9.4.4: Summary of Column Test Results

Period	Status	25 mm ore Extraction %		90 mm ore Extraction %	
		Au	Ag	Au	Ag
14 Days	Leach	64.6	14.3	57.9	9.1
28 Days	Leach	67.4	15.9	61.7	10.8
56 Days	Wash	72.9	17.4	67.5	10.8

Source: Koza, 2012

2.9.5 Process Plant

Process Description

Metallurgical testwork has demonstrated that gold from Mollakara oxide ore is readily recoverable using standard heap leach cyanidation technology. Gold from the transition ore types was found to be less recoverable using this technology. Koza is currently considering heap leaching Mollakara ore at the rate of 6 million tonnes per year using the conceptual process flowsheet shown in Figure 2.9.5.1. Run-of-mine (RoM) ore would be crushed to -90 mm in a single-stage of crushing and then conveyed or truck-hauled to the leach pad. The ore will then be leached with a weak cyanide solution (~400 ppm NaCN) for about 60 days. The column leach tests demonstrated that over 85% of the recoverable gold is extracted within the first 15 to 20 days of leaching, however, in order to scale-up to a commercial operation a 3X factor was applied to allow for inefficiencies normally encountered in full size heap leach operations.

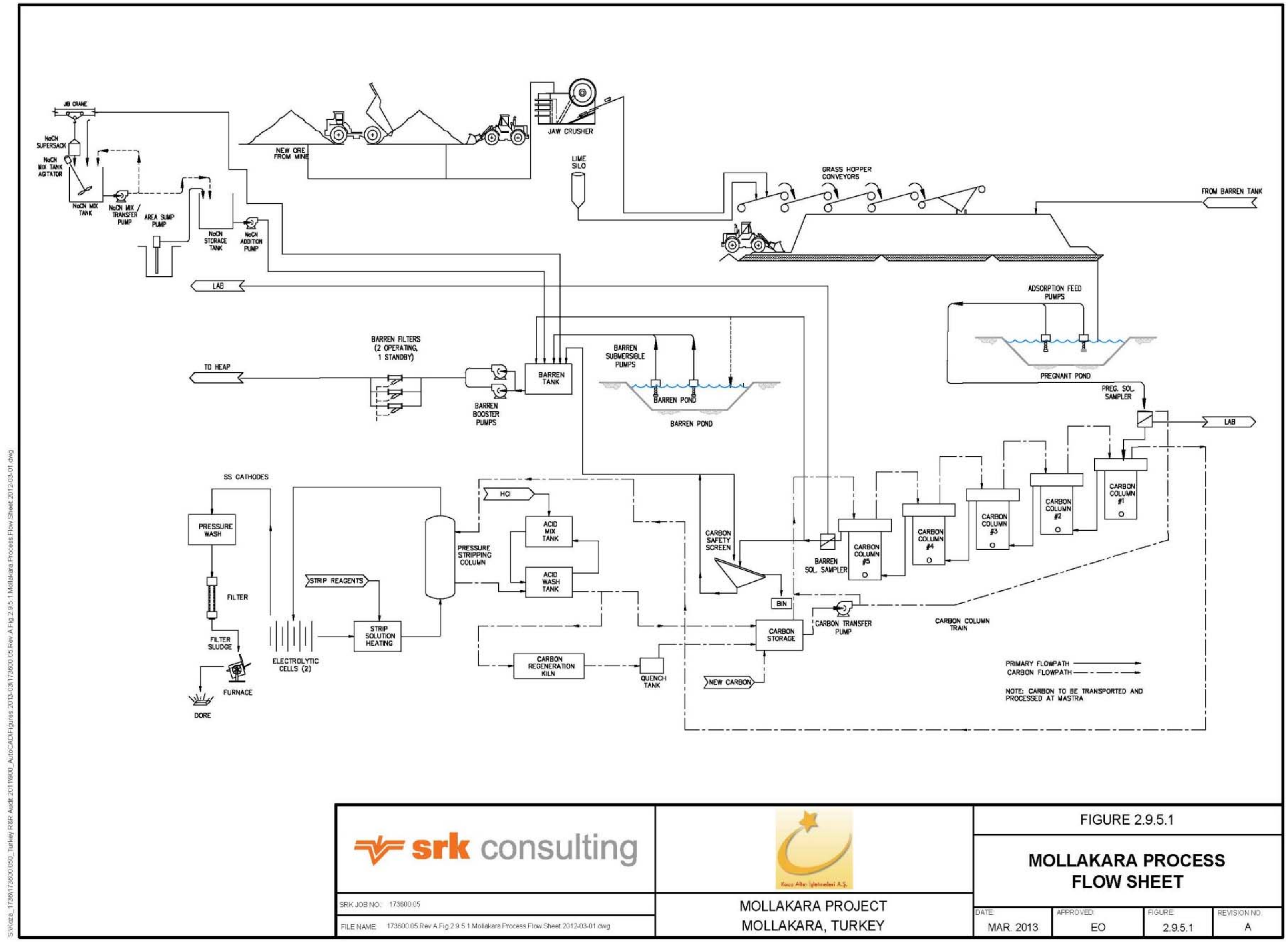


Figure 2.9.5.1: Mollakara Conceptual Process Flow Sheet

Gold contained in the pregnant leach solution will be recovered in a six-stage carbon-in-column (CIC) carbon adsorption circuit where the carbon is moved through the circuit counter-currently to the flow of the pregnant leach solution. It is expected that gold will load onto the carbon to a concentration of about 4,000 – 5,000 g/t Au. The barren solution exiting the CIC circuit will be pumped to the barren pond where the alkalinity and cyanide concentration will be adjusted to the proper levels prior to being recycled back to the heap leach.

The loaded carbon will be trucked to Koza's near-by Mastra Gold Mine where the gold will be stripped from the carbon with a hot caustic solution containing about 3% NaCN. The redissolved gold will be recovered in electrolytic cells to produce a precious metal cathode sludge which will be filtered, retorted to remove mercury and then refined to produce a final doré product. It should be noted that the mercury content of the Mollakara ore is sufficiently high that retorting will be required in the gold recovery circuit to remove the contained mercury prior to refining.

Estimated Recovery

As shown in Table 2.9.5.1, gold extraction from the Mollakara oxide ore averaged 77.3% and gold extraction from the transition ore averaged 40.6% at a crush size of P_{80} -32 mm. Subsequent full-height column tests conducted on a bulk test composite crushed to P_{80} 25 mm and P_{80} 90 mm (Table 2.9.4.4) resulted in 72.9% gold extraction at the P_{80} 25 mm crush size and 67.5% gold extraction at the P_{80} 90 mm crush size. This represents a 5.4% reduction in gold extraction at the coarser crush size.

Table 2.9.5.1: Gold Extraction from Mollakara Oxide and Transition Ores, - 32 mm Crush Size

	Calc Head	Au Extraction	Reagent Consumption		
Transition Ore Type	**Au g/t**	**%**	**NaCN, Kg/t**	**Lime kg/t**	**Cement, Kg/t**
Composite 1	0.42	33.3	1.4	6.6	1.0
Composite 2	1.19	47.9	1.8	7.5	1.0
Average	**0.81**	**40.6**	**1.6**	**7.1**	**1.0**
Oxide Ore					
Composite 3	0.87	85.1	1.0	2.7	1.0
Composite 4	0.51	74.5	1.1	3.3	1.0
Composite 5	0.98	72.4	1.1	4.2	1.0
Composite 6	1.50	77.3	1.2	3.7	1.0
Average	**0.97**	**77.3**	**1.1**	**3.5**	**1.0**

Source: McClelland, 2011

At a P_{80} 90 mm crush size SRK would estimate an overall gold extraction of ~69% on the oxide ore, based on the average oxide ore gold extraction of 77.3% obtained during the McClelland test program at a 32 mm crush size, adjusted down by 5.4.% to allow for reduced gold extraction at the coarser P_{80} 90 mm crush size. An additional 3% reduction in gold extraction is taken to allow for inefficiencies normally encountered in a commercial heap.

Estimated Plant Operating Cost

As shown in Table 2.9.5.2, process plant operating costs are estimated at US$4.98/t and assumed the operation of an "on/off" heap leach at a cost of US$1.69/t of ore. Although a conventional multi-lift heap leach operation is anticipated, the cost of an "on/off" heap leach operation is included in the cost analysis due to the high clay content of the ore, and concern for reduced percolation rates in a multi-lift operation. Load-permeability tests are currently underway to assess the potential for reduced permeability in a multi-lift heap.

Table 2.9.5.2: Estimated Process Plant Operating Costs

Cost Area	US$/t
Labor	0.17
Crushing and Transport to Leach Pad	0.35
Heap Loading/Unloading	1.69
NaCN	1.44
Lime	0.37
LNG	0.02
Electricity	0.03
Maintenance	0.13
Carbon Transport to Mastra	0.33
Elution and Refining at Mastra	0.20
Other (@ 9%)	0.25
Total	**4.98**

Source: Koza, 2012

Labor costs include overhead and allow for a total of 71 employees, with 34 allocated to the process plant, 10 allocated to the laboratory and 27 allocated for maintenance. Crushing and transport to the leach pad will be by contractor, and is based on quotations received for other similar projects. Reagent costs are based on the average reagent consumption obtained during all column tests, however, the average cyanide consumption has been reduced by 50% recognizing that actual plant cyanide consumptions are typically lower than reported from test columns. Gold elution costs are based on Koza's actual cost at other plants. An allowance of US$0.25/t has been provided for other cost categories.

Estimated Process Capital Cost

Koza has estimated the capital cost for the Mollakara heap leach pad and associated process facilities at US$94.3 million. A summary of Koza's capital cost estimate is provided in Table 2.9.5.3.

Table 2.9.5.3: Mollakara Process Capex Summary

Cost Area	Source	US$
Infrastructure	Koza	4,438,978
CIC and Reagents	Koza	3,386,494
Heap Leach Pad	SRK	70,715,370
Indirect Costs	Koza	3,500,000
Subtotal		82,040,842
Contingency	15%	12,306,126
Total Cost		**94,346,968**

Source: Koza/SRK, 2012

2.10 Environmental

Project development, exploration and planning activities have been conducted by Koza after their acquisition from the former owner, Newmont Mining. Exploration and planning for two other prospect areas (Taşlıçay and Çakillitepe) located close to the Mollakara Prospect are conducted in coordination with the Mollakara Project development. The selected mining method is open pit mining and cyanide heap leaching will be used for gold extraction.

2.10.1 EIA and Environmental Permitting

A scoping level environmental assessment was conducted for the Mollakara Project by SRK Turkey in 2009. Study of environmental characteristics of the prospect area and environs covered the information compiled through governmental offices in Ankara, desk-based studies and site surveys. Key findings from environmental assessment and likely environmental sensitivities are:

- The Murat River and its tributaries drain the Mollakara Prospect and its vicinities. The Project area footprint coincides with the main stream of the Murat River. The proximity and required diversion works may create environmental sensitivity which will need to be addressed;
- The project area is classified as pastureland. Hence, the land acquisition procedure will also include change of the legal land use status and permitting from the Ministry of Agriculture; and
- The local hydrogeology might become an environmental sensitivity in the future in the event that the Murat Reservoir Project progresses from the early planning to implementation stage. The Murat Reservoir Project is currently in the early planning stages and the utilization purpose for the reservoir is not clear yet.

The environmental baseline assessment and EIA studies for Mollakara were initiated in December 2009. The specialized baseline studies including field surveys, periodic sampling and monitoring have been started for baseline water quality, hydrology, hydrogeology, geochemistry, terrestrial and aquatic flora and fauna, baseline air quality, and noise. The Mollakara Heap Leach Pad design study has been completed by SRK, and the legal EIA procedure for Mollakara was initiated in 2011. The EIA positive certificate was received on August 23, 2012.

3 Conclusions and Recommendations

3.1.1 Geology and Resources

Should Koza conduct additional drilling for resources at Mollakara, SRK recommends that Koza monitor the decreasing grade trend of SE44 and HiSilK2 and contact the laboratory should it continue. Koza should also include silver standards in its QA/QC program at this project. In reference to duplicates, SRK recommends discontinuing the insertion of core duplicates and adding preparation and pulp duplicates to its QA/QC program as well as sending a subset of pulp duplicates to a second laboratory to verify analysis. Should Koza drill a new area that has significantly different geology, mineralization texture or type, SRK recommends using core duplicates to test for nugget effect, determine grind size for analysis and to determine adequate sample submission size. Once these factors are assessed, then Koza can discontinue core duplicates at the project. SRK also recommends that Koza assess failures in the context of what type of failure occurred and how many failures occurred in each batch. If all of the QA/QC samples failed in the batch, then the entire batch should be reanalyzed. However, if only one QA/QC sample failed, then the failed QA/QC sample plus three to four samples in sequence on either side of the failure should be reanalyzed.

SRK also recommends that Koza consider the following performance gates for CRMs:

- If one analysis is outside of ±2 standard deviations it is a warning;
- Two or more consecutive analyses outside of ±2 standard deviations is a failure;
- If an analysis is outside ±3 standard deviations it is a failure if ±3 standard deviations does not exceed ±10% of the mean; and
- If the ±3 standard deviations exceed ±10% of the mean, then ±5 to ±10% should be used.

Ore Research & Exploration (OREAS), who manufactures CRMs, recommends using these performance gates and has started printing this information on CRM certificates as part of a guide for use of the CRM. ALS Global uses ±3 standard deviations during analysis as a performance gate for internal CRMs (ALS Global, 2012). Koza is using a more restrictive performance gate that may result in unnecessary failures.

SRK recommends that in future resource estimations that Koza incorporate lithological and structure information into the grade shell.

3.1.2 Mining and Reserves

Mollakara will be Koza's second heap leach operation and should benefit from experiences gathered at Himmetdede. The combination of seasonal mining, water management, extended transportation of ore to the heap pad and relatively low grade, will challenge mining operations at Mollakara. Care will need to be taken that surface and groundwater interfacing with the pit and waste dump does not enter the Murat River giving rise to social and environmental concerns. Blasting and material handling will need to be considered carefully given the potential clay content of the ore and potential effect on percolation. SRK recommends that material properties such as moisture content, swell and clay content be defined from a mining perspective and storm water management plan implemented for the site.

Because the global stability of the main pit wall is dependent on groundwater draw down to get a factor of safety higher than 1.27, SRK recommends that local stability analysis be carried out for the pit designs. The change in lithology half way up the slope may cause problems in the top portion of the wall particularly as the nose of the ridgeline is cut.

The location of the heap leach pad needs to be defined so accurate mine costing can be updated. This will require a 2015 field program to test each potential site for geotechnical stability as at the pad there are old failure surfaces common in the area. The situation may lead to pad movement and eventual loss if the ground moves or creeps significantly.

AMD and ARD testing (if not done so already) should continue. Of particular concern is the location of the waste dump and vicinity to the river. There should also be infrastructure studies completed on river protection structures, bridges and flood mitigation. All this ties into the requirement for an integrated storm water management plan.

3.1.3 Metallurgy and Process

- At a P_{80} 90 mm crush size, SRK estimates an overall gold extraction of ~69% on the oxide ore;
- The high clay fines content of the oxide ore may present material handling difficulties during commercial crushing, agglomerating and heap leaching. In particular, difficulties can be expected if the high clay ore has a significant moisture content when fed to the crushing plant;
- The transitional ore type material was not as amenable to simulated heap leach cyanidation, and gold extraction was more sensitive to feed size;
- Process plant operating costs are estimated at US$4.98/t; and
- Koza has estimated the capital cost for the Mollakara heap leach pad and associated process facilities at US$94.3 million.

3.1.4 Environmental

Key findings from environmental assessment and likely environmental sensitivities are:

- The Project area footprint coincides with the main stream of the Murat River. The proximity and required diversion works may create environmental sensitivity which will need to be addressed; and
- The local hydrogeology might become an environmental sensitivity in the future in the event that the Murat Reservoir Project progresses from the early planning to implementation stage.

The Mollakara Heap Leach Pad design study has been completed by SRK, and the legal EIA procedure for Mollakara was initiated in 2011. The EIA positive certificate was received on August 23, 2012.

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5 Glossary

5.1 Mineral Resources and Reserves

The JORC Code 2012 was used in this report to define resources and reserves.

A 'Mineral Resource' is a concentration or occurrence of material of intrinsic economic interest in or on the Earth's crust in such form, quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories.

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which tonnage, grade and mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes which may be limited or of uncertain quality and reliability.

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.

A 'Measured Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes. The locations are spaced closely enough to confirm geological and grade continuity.

5.2 Glossary of Terms

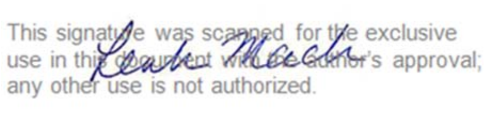
Table 5.2.1: Glossary

Term	Definition
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All other expenditures not classified as operating costs.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing	Initial process of reducing ore particle size to render it more amenable for further processing.
Cutoff Grade	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution	Waste, which is unavoidably mined with ore.
Dip	Angle of inclination of a geological feature/rock from the horizontal.
Fault	The surface of a fracture along which movement has occurred.
Flitch	Mining horizon within a bench. Basis of Selective Mining Unit and excavator dig depth.
Footwall	The underlying side of an orebody or stope.
Grade	The measure of concentration of gold within mineralized rock.
Haulage	A horizontal underground excavation which is used to transport mined ore.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Milling	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mining Assets	The Material Properties and Significant Exploration Properties.
SAG Mill	Semi-autogenous grinding mill, a rotating mill similar to a ball mill that utilizes the feed rock material as the primary grinding media.
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Sill	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Spigotted	Tap/valve for controlling the release of tailings.
Stope	Underground void created by mining.
Strike	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide	A sulfur bearing mineral.
Tailings	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening	The process of concentrating solid particles in suspension.
Variogram	A statistical representation of the characteristics (usually grade).

6 Date and Signature Page

Signed on this 31st Day of January, 2015.

Endorsed by CPs:



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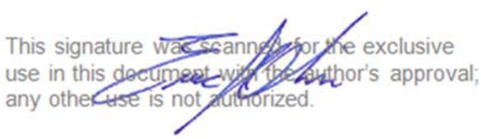

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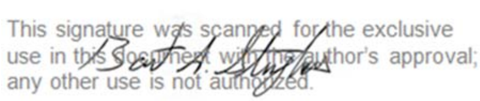

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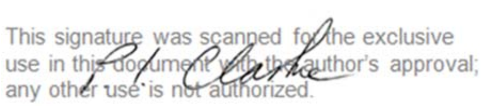

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All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted industry practices.

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